MINERALS

ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT

Final Report
TASK 1A - VBM Design Guidelines
Chapter 2 - Data Collection Systems
Volume I - Report

by

Inco Limited, Thompson Division
Thompson, Manitoba

March 1990

Canada

Manitoba

CANADA - MANITOBA

MINERAL DEVELOPMENT AGREEMENT

RESEARCH & DEVELOPMENT IN MANITOBA MINES

Work on the project was conducted under the auspices of Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources Canada under the Canada/Manitoba Mineral Development Agreement.

DSS File NO. 14SQ.23440-4-9147-3
DSS Contract Serial No. OSQ84.00403
CANMET Project NO. 4-9147-3
CANMET Scientific Authority: Y.S. Yu

Final Report
TASK 1A - VBM Design Guidelines
Chapter 2 - Data Collection Systems
Volume I - Report

by

Inco Limited, Thompson Division
Thompson, Manitoba

March 1990

96 pp

MICROMEDIA

CANADA - MANITOBA
MINERAL DEVELOPMENT AGREEMENT
RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

Chapter 2

Data Collection Systems

This chapter has been prepared as part of the

TASK 1A
VERTICAL BLOCK MINING DESIGN GUIDELINES PROJECT
of the
Canada/Manitoba Mineral Development Agreement

by

G. Greer, R. Platford, R. Proulx

INCO Limited, Thompson Division, Manitoba

DSS File No.: 14SQ.23440-4-9147-3
CANMET Project No.: 4-9147-3
CANMET Scientific Authority: Y.S. Yu

90-05668/2

CANADA - MANITOBA

MINERAL DEVELOPMENT AGREEMENT

RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

The Vertical Block Mining (VBM) Design Guidelines consists seven chapters, published separately. The seven chapters are:

Summary

1. **Cut and Fill Mining**
2. **Data Collection Systems**
3. **VBM Stope Design**
4. **Drilling and Blasting**
5. **Support Requirements**
6. **VBM Stoping Procedures**

Reference to this chapter should be quoted as follows: "**Vertical Block Mining Design Guidelines Chapter 2 - Data Collection Systems**"; Canada/Manitoba Mineral Development Sector B - Mining; DSS File No. 14SQ.23440-4-9147-3; CANMET Project No. 4-9147-3; March 1990

EXECUTIVE SUMMARY

Chapter 2: Data Collection Systems

The development and maint ance of a well structured data collection system is of fundamental importance to a mining operations, regardless of the mining method employed. At Thompson Mine, a data collection system was developed based on the case histories from the mining of 126 vertical crater retreat blocks.

The database was separated into two sections: geotechnical and mine operational data. The geotechnical database included categorizations such as geology, structure, strength and rock mass quality. The concept of stress and the effect of the redistribution of stresses through the use of visual monitoring and field instrumentation techniques was included in this geotechnical database.

The mine operational database was separated into categories such as stope geometry, development, drilling, blasting, and mucking. The mining efficiency, measured from the combination of parameters from these groupings, was subsequently derived from the mine operational database.

Data collection serves three purposes. Firstly, data assists in understanding the actual situation so that proper evaluations can be made. Secondly, the proper evaluation of data will form the basis for actions and decisions. Too often decisions are made on judgement, based on opinions and experiences with no records for backing. Thirdly, a database allows current data to be compared with historical data to determine whether or not a process is proceeding as normal, or whether the process has actually changed.

The effective use of a data collection system can be summarized best by the following list of activities:

- comparison of individual mining and geotechnical parameters;
- analysis of operating parameters in order to determine levels of control;
- indication of potential problems in current mining areas;
- development of empirical relationships for the purpose of future mine design.

SOMMAIRE

Chapitre 2: Système de récolte de données

Le développement et la mise à jour d'un système de récolte de données bien structuré ont une importance fondamentale dans une opération minière, peu importe la méthode de minage employée. Pour la mine Thompson, un système de récolte de données a été développé et appliqué sur 126 blocs minés antérieurement par la méthode des cratères par retrait vertical (VCR).

La base de données a été séparée en deux parties: les données géotechniques et les données opérationnelles de la mine. Les données géotechniques incluent des catégories telles que géologiques, structurales, sur la résistance et la qualité de la masse rocheuse. Les concepts de contraintes et l'effet de la redistribution des contraintes après percement ont été inclus dans la base de données géotechniques à l'aide d'inspections visuelles et des techniques d'instrumentation.

La base de données opérationnelles de la mine a été séparée en catégories tels que la géométrie des chantiers, le développement, le forage, le sautage et le marrinage. La mesure d'efficacité du minage, mesurée à partir de la combinaison de paramètres des différents groupes, provient de la base de données opérationelles de la mine.

La récolte de données est utilisée pour trois buts. Premièrement, les données permettent de comprendre la situation actuelle et rendent possible des évaluations adéquates. Deuxièmement, ces évaluations des données vont former une base fiable sur laquelle on peut s'appuyer pour prendre des décisions et des actions. Troisièmement, la base de données récoltées peuvent être comparées avec les données antérieures afin de déterminer si l'opération se déroule normalement ou non ou si la méthode d'opération a subit des changements.

L'utilisation efficace d'un système de récoltes de données peut être résumée par la liste d'activités suivantes:

- la comparaison entre le minage individuel et les paramètres géotechniques.
- l'analyse des paramètres d'opération afin de déterminer le niveau de contrôle.
- l'indication de problèmes potentiels dans les régions de minage actif.
- le développement de relations empiriques pour établir les futurs designs de mine.

Table of Contents

1.0 INTRODUCTION 1

2.0 GEOTECHNICAL DATA COLLECTION 2
- 2.1 Introduction 2
- 2.2 Geology 3
- 2.3 Structure 4
- 2.4 Strength 5
- 2.5 Rock Mass Quality 7
- 2.6 Stress 9
- 2.7 Visual Monitoring 10
  - 2.7.1 Introduction 10
  - 2.7.2 Section Sketching 10
  - 2.7.3 Blasthole Surveys 11
  - 2.7.4 Drawpoint Monitoring 11
  - 2.7.5 Reporting of Results 11
- 2.8 Field Instrumentation 12
  - 2.8.1 Introduction 12
  - 2.8.2 Ground Movement Instrumentation 12
  - 2.8.3 Local Stress Measurement 12
  - 2.8.4 Stress Measurement at Thompson Mine 13

3.0 MINE OPERATIONAL DATA COLLECTION 13
- 3.1 Introduction 13
- 3.2 Stope Geometry 14
- 3.3 Stope Development 15
- 3.4 Production Drilling 15
- 3.5 Production Blasting 16
- 3.6 Production Mucking 17
- 3.7 Mining Efficiency 17
  - 3.7.1 Dilution 17
  - 3.7.2 Recovery 18
  - 3.7.3 Productivity 18
  - 3.7.4 Cost Efficiency 19

4.0 SUMMARY 19

5.0 REFERENCES 21

APPENDICES A - F

List of Figures

Fig. No.	Description	
2.1	Thompson Mine Longitudinal	22
2.2	Hanging Wall Rock Types	23
2.3	Ore Zone Rock Types	24
2.4	Foot Wall Rock Types	25
2.5	Plan of Regional Structure at Thompson Mine	26
2.6	Dip of VBM Blocks	27
2.7	Stereonet - Nose Zone	28
2.8	Stereonet - 2 Shear	29
2.9	Stereonet - 3 Shear	30
2.10	Stereonet - 1A Zone	31
2.11	Stereonet - 1B Zone	32
2.12	Stereonet - 1C Zone	33
2.13	Stereonet - Thompson Mine	34
2.14	Joint Set Orientation By Ore Zone Thompson Mine	35
2.15	Rock Strength Description	36
2.16	Estimation of Rock Mass Strength	37
2.17	Estimation of Rock Mass Strength	38
2.18	Estimation of Rock Mass Strength	39
2.19	Estimation of Rock Mass Strength	40
2.20	Estimation of Rock Mass Strength	41
2.21	Estimation of Rock Mass Strength	42
2.22	Estimation of Rock Mass Strength	43
2.23	Estimation of Rock Mass Strength	44
2.24	Estimation of Rock Mass Strength	45
2.25	Estimation of Rock Mass Strength	46
2.26	Classification Rock Mass Properties	47
2.27	Uniaxial Compressive Strength Tests	48
2.28	Relation Between Stand-Up Time, Excavation Span and CSIR Classification	49
2.29	Rock Mass Rating Support Classification	50
2.30	Hangingwall Quality	51
2.31	Ore Quality	52
2.32	Footwall Quality	53
2.33	Rock Mass Quality in Open Stope Backs	54
2.34	Rock Mass Quality in Hanging Walls	55
2.35	Results of Canmet Stress Measurements	56
2.36	Stress Orientation VS. Geological Structures	57
2.37	Direction of Compression During Tectonic Deformation	58
2.38	Measured Vertical Stresses Compared with World Data	59
2.39	Horizontal/Vertical Stress Compared with World Data	60
2.40	Estimated Vertical Stress Gradient	61

Fig. No.	Description	
2.41	Estimated K-Ratio Gradient	62
2.42	Stress State	63
2.43	Typical Failure Modes	64
2.44	Scaling Effort	65
2.45	Support Type	66
2.46	Support Load	67
2.47	Observation Sheet - Openings	68
2.48	Blasthole Condition	69
2.49	ITH Observation	70
2.50	Typical Drawpoint Recording Sheet	71
2.51	Standard VBM Showing Stages of Mining	72
2.52	VBM Block VS. Stope	73
2.53	Retreat VBM - Longitudinal Mining	74
2.54	Location of VBM Blocks in Thompson Mine	75
2.55	Height of VBM Blocks at Thompson Mine	76
2.56	VBM Block Widths at Thompson Mine	77
2.57	Length of VBM Blocks at Thompson Mine	78
2.58	Transverse Block Drawpoint Spacing	79
2.59	Extraction Drift Pillar Width	80
2.60	4.5" Footage Drilled Per Shift	81
2.61	6.5" Footage Drilled Per Shift	82
2.62	4.5" Tons Broken/Foot Drilled	83
2.63	6.5" Tons Broken/Foot Drilled	84
2.64	4.5" Powder Factor	85
2.65	6.5" Powder Factor	86
2.66	Tons Mucked Per Shift	87
2.67	Drawpoint to Orepass Haul Distance	88

List of Appendices

APPENDIX A GLOSSARY OF ITEMS COLLECTED IN PROJECT DATABASE

APPENDIX B STOPE DATA FILES

APPENDIX C CASE STUDY - DEVELOPMENT OF A VERTICAL CRATER RETREAT MINE DATABASE AT INCO'S THOMPSON MINE

APPENDIX D METHODS OF LINE SURVEYS

APPENDIX E CASE HISTORY FOR SECTION SKETCHING

APPENDIX F CASE HISTORY - FIELD INSTRUMENTATION

1.0 INTRODUCTION

INCO's Thompson Mine is a primary producer of nickel ore originating from both open pit and underground operations. The mine crown pillar is currently being extracted using open pit mining methods, while the ore zone below 400 level is being mined using conventional underground extraction techniques. Production from the mine averages 5443 tonnes (6000 tons) per day, with the open pit producing a comparable tonnage.

The Thompson Mine ore zone extends over a strike length of 5.6 km (3.5 miles). It is located in a schist zone conformable with the local formation. The ore pinches and swells both along strike and down dip, with further irregularities due to local tension cracks into the wall rocks. Ore zone widths range from the minimum economic mining width (approximately 1.5 m or 5 ft) up to a maximum of 45 m (150 ft).

The underground mine is serviced by two shafts, as shown in Figure 2.1. The T1 shaft, at a depth of 1350 m (4427 ft), is located at the south end of the ore zone and serves as the main production shaft. The T3 shaft, located two miles north of T1, operates as a service shaft for the northern half of the ore zone to a depth of 795 m (2607 ft). A ramp is currently being developed at the T3 shaft from the 2400L to the 3600L horizon for development of the ore zone extending between these two levels.

Production from the Thompson Mine commenced in 1961. From that time until approximately 1980, the mine was primarily a cut and fill operation. In the early 1980's, vertical block mining was introduced and a major mining method conversion effort commenced. The conversion has progressed to a stage whereby vertical block mining contributes upwards of 80% of the present underground mine production.

Vertical block mining, or VBM, is a name applied to the open stoping methods used at Thompson Mine. Other names that are used synonymously with VBM are vertical crater retreat (VCR) and vertical retreat mining (VRM), both forms of open stoping with minor variations from VBM.

In order to properly establish a set of vertical block mining design guidelines as a requirement of this CANMET project, a representative cross-section of historical data from VBM areas is required. As such, a database system was established and completed for 126 blocks mined at the Thompson Mine. These blocks were extracted using VBM techniques between the years 1981 to 1987. A total of 106 parameters were collected and tabulated for each block. The parameters are listed and described in Appendix A.

The data was then entered on a PC-based spreadsheet system (Microsoft Excel), with one spreadsheet being identified per block. Summary spreadsheets were established in order to extract and summarize specific parameters from each block spreadsheet through a built-in linking system. This system was semi-automated in that a change in the value of a parameter in an individual block spreadsheet automatically generated a change in that same value in the summary sheet. Copies of the data files for each of the 126 blocks are included in Appendix B.

The data that was generated on the summary spreadsheets has been used to create histograms and scatter plots of the selected parameters. These diagrams form the basis of this and the following chapters of the report. By no means do these diagrams represent all of the possible graphs that can be generated from this database, but instead illustrate the usefulness of such a database collection and analysis system. Appendix C contains a technical paper illustrating a case history whereby the database system was used to analyze the stability of the Thompson Mine block hanging walls.

For the purpose of describing and assessing the project data collection system, this particular chapter has been sub-divided into two sections: geotechnical data collection and mine operational data collection.

2.0 GEOTECHNICAL DATA COLLECTION

2.1 Introduction

Geotechnical data collection was defined for the purpose of this project as consisting of the following items:

- rock mass properties
- stress fields
- visual monitoring and instrumentation

A fundamental purpose for the collection of this type of data is for the calibration of models, whether they be numerical, empirical or analytical. This data was collected for two separate purposes for this project. First, it was required for the calibration of empirical models designed to predict stope stability (as covered in Chapter 3 - Stope Design). Second, it was necessary for the development of a strong geotechnical information base that could be used to describe and analyze the Thompson Mine geotechnical features.

In the situation whereby a mine operates without the services of a ground control department, as in the case at Thompson Mine, the question arises as to who should ultimately be responsible for the collection and reduction of this data. The first component, rock mass properties, should realistically be assembled by members of the mine geology department. The mine geologists are required as part of their regular duties to map headings and log diamond drill core, and as a result are exposed to items such as fracture frequency, joint alteration and infilling on a daily basis.

The interpretation and transformation of this data into a form suitable for further calculation should be left to the responsibility of a geotechnical or mining engineer. These engineers have the technical background necessary to properly perform this type of evaluation.

It is suggested that the determination of magnitude and direction of the inherent stress regime be left to individuals from the geotechnical consulting industry. Most of these consulting firms have in-house experts who can perform this type of work with greater accuracy and at a lesser cost than the average mining operation.

Numerical modelling based on the results of the field stress measurements can be carried out by engineering personnel at the minesite under the direction of the geotechnical consultants. The responsibility of the consulting firms initially would be to recommend and supply the appropriate numerical models for the given situation, as well as to instruct industry personnel on the proper use and interpretation of these models.

Visual monitoring and field instrumentation is generally a function of the mine engineering group. Individuals from this department are regularly underground and have the opportunity to design, install and subsequently monitor these programs. Input from members of the geotechnical consulting industry may again be required for the interpretation of data resulting from these field instrumentation systems.

In evaluating the geotechnical data collection approach for this project, most of the responsibility was delegated as outlined above. All virgin stress data was compiled from an open pit rock mechanics study performed by Steffen Robertson and Kirsten Inc. at Thompson Mine in 1984. Numerical modelling was performed primarily by individuals from the mine engineering and mine research groups. Visual monitoring and instrumentation was the ultimate responsibility of the mine research group, which at the Thompson mine is an extension of the engineering department. Collection of rock mass properties, however, deviated from the above suggested standard in that the engineering group was responsible for conducting a majority of the scanline surveys. Some input from the geology department was required in that they did perform some scanline survey work as well as contribute rock quality designation data from diamond drill core.

The remaining portion of this chapter is presented in a logical sequence. The following three sections assess the results of the geotechnical data collection involving geology, structure and strength, respectively. These three entities are then grouped together in order to present the next section, rock mass quality.

Following this, the particulars of ground stress at Thompson Mine are presented, particularly regional stress as opposed to local stress. The final two sections identify the methodology used at Thompson Mine in order to monitor the effect of ground stresses given the inherent rock mass quality.

2.2 Geology

In an operating mine, a definition of the term geology can best be simplified as a description of an orebody geometry and composition. The collection of this type of information is the ultimate responsibility of a mine geology department. Most of this material can be provided through underground mapping exercises and from diamond drilling programs.

The collection and organization of this geology information forms the framework for many aspects of the ultimate mine design. Results from the other geotechnical data analysis procedures, such as rock strength and structure, can be integrated with the geology data in order to provide an overall rock mass quality.

The primary rock types that are found in the VBM stope hanging walls, ore zones, and footwalls at Thompson Mine are illustrated in Figures 2.2, 2.3 and 2.4. These histograms were generated from the project database.

In addressing each of the histograms individually, the predominant rock type found in the Thompson Mine hanging wall is a biotite schist, as shown in Figure 2.2. The thickness of this schist in general ranges from 1.5 to 7.5 m (5 to 25 ft), and the degree of schistosity plays a major role in wall stability. Other rock types occasionally found on the immediate hanging walls of some stopes include peridotite and quartzite.

The stope backs, by in large, are composed of a sulphide matrix. This matrix consists of a massive sulphide containing varying degrees of brecciated inclusions. Mineralized schist and massive sulphides are also common to stope backs.

The footwall rock types are illustrated in Figure 2.4. Silicious schist and quartzite are most common in the footwall. Peridotite walls are sometimes observed, primarily in the 3 shear ore zone.

2.3 Structure

In the discussion of properties pertaining to rock mass structure, the scope of this term is separated into two categories, those being regional structure and local structure.

Regional structure is defined as the location and orientation of the prominent structural features of an area, such as in major shear and fault zones. Local structure, on the other hand, is best defined as the description of the jointing and fracturing within a given zone as defined by regional structure.

The regional structure for the Thompson Mine is illustrated in Figure 2.5. For analytical purposes pertaining to the evaluation of vertical block mining, the orebody was divided into 6 distinct ore zones, these being the 1C, 1B, 1A, nose, 2-shear and 3-shear zones, each of which possess their own local structural characteristics.

Figure 2.6 illustrates a histogram representing the average dips of the VBM stopes included in the database system. The average dip of the ore zone, as per the data from the histogram, is 69 degrees, which by in large represents the average dip of the entire Thompson Mine ore zone. Orebody dips do range, however, from 45 degrees to 90 degrees, the vertical walls occurring primarily in the 2 shear ore zone.

In determining local structure, one method used to identify the discontinuity (joint) sets is accomplished through performing detailed scanline surveys (also referred to as line surveys). Detailed line surveys as described in Appendix D are extremely time consuming and involve detailed data collection for each joint intersected by the baseline. The revised scanline survey method found in Appendix D streamlines the data collection by identifying the frequency of joint sets and recording the orientation and surface conditions of the particular set only once. This method allows information to be collected quickly as compared with the detailed scanline surveys.

Scanline surveys were carried out in each of the six ore zones at Thompson Mine in order to determine potential trends in joint orientation. Dip directions and angles were entered into a PC-based stereonet program, which produced the stereonet plots shown in Figures 2.7 to 2.12. These stereonets represent contour plots in lower hemisphere projection. An overall mine stereonet was produces by combining data from all six ore zones, as illustrated in Figure 2.13. In this manner occurrences and orientations of joint sets were identified within each zone and compared with regional trends. A summary of the orientations of joint sets within each zone is presented in Figure 2.14.

Four major joint sets can be identified on each stereonet, with the exception of the stereonet representing the jointing in the nose zone, where it is apparent that there is an insufficient amount of data to properly identify all of the major sets. Two additional random joint sets were found in the 2 shear and 1C stereonet plots.

In general, the jointing is predominantly steeply dipping. Joint set 1 represents the foliation surfaces associated with the orebody. Joint set 2 is shallow dipping and in general perpendicular to the foliation set. For all due purposes, joint sets 3 and 4 can be grouped as one set. These sets are steeply dipping and approximately perpendicular to the foliation set. They also parallel the regional MacKenzie dyke swarm. Joint set 5, a random set, is found in the 2 shear zone and is also steeply dipping, with an orientation between the foliation set and joint set 3. Joint set 6, again random, is found predominantly in the 1C zone, and is perpendicular to the foliation with an intermediate angle.

The random nature of joint set 5 is most likely due to the unusual orientation of the 2 shear ore zone with respect to the other ore zones at Thompson Mine. In all of the plots, the statistical contouring of the stereonet was beneficial in suppressing the orientation of individual surfaces to a common joint set.

The strengths and weaknesses of collecting structural data from diamond drill core versus underground scanline surveys can be argued at length. The principal advantage of the scanline survey technique is the resulting detailed information with respect to joint set orientation and spacing. Obtaining data from drill core, on the other hand, is advantageous in that it is far less time consuming and it also provides a source of data from areas of the orebody which are physically inaccessible.

For the purpose of this project, data from both sources was collected and combined, with the preference being placed on the data from the scanline surveys. These surveys were performed in underground headings which were in close proximity to vertical block mining areas. Drill core interpretation was used in areas where either physical access was denied or where time constraints prevented initiation of a scanline survey program. In some areas, a combination of this data was used in the determination of local structure.

2.4 Strength

The discussion of rock strength involves three parameters.

i) strength of intact rock
ii) strength along discontinuities
iii) strength of the overall rock mass

Intact rock strength is evaluated from the results of point load, uniaxial compression, or tensile strength testing. The method most often used and the one that provides a basic means of classifying rock, is the uniaxial compressive strength test. This test is normally performed under laboratory conditions using selected core samples. In order to assess the strength of the rock in the field, point load strength apparatus and the Schmidt hammer can be used. Though both apparatus have their suitable applications, they are difficult to use because of limited access in the underground environment. The best overall approach involves the development of a simple field strength estimating system that can be used in conjunction with laboratory results. An example of this is shown in Figure 2.15.

The shear strength along discontinuities is controlled by the surface condition of the joint. The most commonly used approach to establish shear strength is the Barton and Bandis (2) (1982) equation.

$$\theta peak = JRC \log \frac{(JCS)}{\sigma_n} + \theta b$$

$\theta peak$ = Peak Friction Angle
JRC = Joint Roughness Coefficient
JCS = Joint Wall Compressive Strength
θb = Base Friction Angle
σ_n = Effective Normal Stress on Joint

Both JRC and JCS can be obtained from field data while θb is determined from direct shear tests on

saw cut samples.

Jointed rock masses are comprised of interlocking blocks of intact rock separated by discontinuity surfaces. The strength of a rock mass will depend on the strength of the intact pieces and on their freedom of movement which, in turn, depends on the number , orientation, spacing and shear strength of the discontinuities. Hoek and Brown (8) (1980) have developed an approximate method for estimating the strength of such rock masses. The method relates the major principal effective stress at failure (σ_1') and the minor principal effective or confining stress (σ_3') to the uniaxial compressive strength of intact rock (σ_c) and two constants, m and s.

$$\sigma_1' = \sigma_3' + (m\sigma_c\sigma_3' + s\sigma_c^2)^{1/2}$$

To establish values for the constants m and s, Hoek and Brown (8) (1980) have used the rock mass classification system devised by Barton, Lien and Lunde (1) (1974) and Bieniawski (4) (1976). Values for m and s have been set against the various classes of rock mass quality and the ratings that occur within the systems. The constants in m and s, in conjunction with the uniaxial compressive strength value, can be input into the Hoek and Brown failure criteria and used to predict approximate failure envelopes.

The results for Thompson Mine are illustrated in Figures 2.16 to 2.24. In order to examine the effect of the inherent variability of intact rock strength on overall rock mass strength, three failure envelopes are shown on each figure. The lower envelope represents the approximate rock mass strength when the minimum value of uniaxial compressive strength measured is used as input; the central envelope when the mean strength is input; and the upper envelope when the maximum compressive strength is input. All envelopes represent peak strength conditions of in-place rock masses and assume that the rock mass is generally isotropic.

In rock masses with strong foliation and consequently planes of weakness, as in the case at Thompson Mine, the overall strength will depend on the orientation of stress. Hoek has developed a computer program that will allow the strength on isotropy of schists to be predicted for different inclinations of the major principal stress axis to the foliation surface. The results are shown in Figure 2.25.

Rock mass deformability is important in assessing how the overall rock mass responds to stresses generated by mining. The rock mass deformation moduli are reported in terms of the elastic deformation moduli: Young's Modulus (E) and Poisson's Ratio (ν). Joint orientation, spacing and continuity all contribute to influence deformation. Direct measurement of deformability has to be estimated from the results of laboratory tests on intact samples, from observation of the rock mass, and from empirical relationships.

At Thompson Mine, values for intact uniaxial compressive strength, tensile strength and deformation modulus measured in the laboratory were very consistent for each major rock type. The results obtained indicate that the rock mass in the mine can be divided into five groups. The groups comprise of:

Group 1 - Biotite gneiss, amphibolite, unleached skarn, hanging wall quartzite and footwall quartzite.

Group 2 - Hanging wall schist

Group 3 - Ore rocks

Group 4 - Footwall schists

Group 5 - Leached skarn

These groups have been summarized graphically in a plot of uniaxial compressive strength versus deformation (Figure 2.26). The table in Figure 2.27 summarizes the numerical values representing the uniaxial compressive strengths of the various rock types from the mine.

2.5 Rock Mass Quality

A rock mass classification system quantifies those properties or features of the rock mass considered likely to influence its behaviour, and combines these individual values into one overall classification rating for the rock mass. Classification systems developed in recent years make use of measurable parameters in an attempt to minimize judgmental bias. The suggested method of rock mass description developed by the Working Party of the International Society for Rock Mechanics, for example, lots the following parameters:

- number of joint sets
- orientation
- spacing (block size)
- condition of joint walls (intact rock strength)
- joint roughness
- joint persistence
- joint opening (aperture)
- water seepage

All of these parameters can either be directly measured or scaled against carefully calibrated descriptive scales. Modern tunnelling classifications use similar parameters to develop an overall rating of the rock mass. Two such systems are in general use.

The CSIR Rock Mass Rating [Bieniawski, (3) (1973)] is based on applying a series of weighted ratings to the various rock mass parameters. The overall Rock Mass Rating (RMR) is produced through addition of individual ratings. The RMR value varies linearly and relates to the qualitative aspects of rock, from 0 for very poor rock to 100 for very good rock.

To obtain an estimate of the expected ground condition, the RMR value is interpreted in terms of stand-up times for the particular dimensions of the opening being assessed. The relationship between RMR, span and stand-up time is presented in Figure 2.28.

A similar concept is used in the NGI System [Barton, et al., (1) (1974)]. In this system, the structure of the rock mass, the frictional characteristics of fractures, and stresses surrounding the opening are individually assessed and combined to give the Tunnelling Quality Index (Q). This index has been related to the span of the opening that will stand unsupported as illustrated in Figure 2.29.

Though both systems are applicable, a significant background of experience has been built up with the N.G.I. system. Hoek and Brown (8) (1980) use the N.G.I. system for evaluation of rock mass strength. Mathews et al (9) (1981) have reported success in using it for assessing open stope design. Potvin et al (11)

(1988) have subsequently modified Mathews' application and presently use the method for routine open stope design calculations.

The N.G.I. system can be represented by the following equation:

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF}$$

Where:

RQD = denotes the rock quality designation
Jn = the joint set number
Jr = the joint roughness number
Ja = the joint alteration number
Jw = the joint water reduction factor
SRF = the stress reduction factor

The Quality Index (Q) system assesses and combines the structure of the rock mass, the frictional characteristics of fractures, and the stresses surrounding the opening. The Q value varies from 0.001 to 1000 on a logarithmic scale and is related to qualitative rock mass assessments. A value of 0.001 indicates exceptionally poor rock mass quality, while a value of 1000 would indicate a very good rock mass quality.

The physical significance of the index can be explained in that the terms RQD/Jn represents in principle the average block size of the mass; Jr/Ja represents the average total shearing strength of the joints; Jw represents both the relative potential head of water on the support and the relative potential of the water eroding the joint infilling; and SRF represents both the structurally controlled kinematic possibility of dislodgement of blocks of ground, and the concentrated stress related confinement, fracturing, squeezing and swelling of the excavation walls.

The N.G.I. system of rock mass classification has been applied to the major rock types within the Thompson Mine. Values for RQD, Jn, Jr, and Ja were obtained from scanline surveys. The quotient (Jw/SRF), which accounts for the effect of stress and ground water in the rock mass, was eliminated since it's application is suitable more for tunnelling and subsequently does not represent the stress conditions induced around open stopes.

Values of Q were determined for the hanging wall, ore zone and footwall for each stope in the Thompson Mine database, as shown in Figure 2.30, 2.31 and 2.32. The Q values for the hanging wall ranged from a low of 3 to a high of 14. Most hanging walls, however, maintained an average rock mass quality of between 9 and 11. The primary reason for the variance in the values can be related to the varying degrees of schistosity of the walls.

The Q values for the ore zone display a slightly tighter distribution, as shown in Figure 2.31. The range of values was from 6 to 11, with the average being approximately 9.

The distribution of rock mass qualities for the footwall appear to parallel those of the hanging wall, with a range of 3 to 14 and an average of approximately 10. The reason for the discrete distribution of points on the hanging wall and footwall histograms can be explained by the method used to determine the rock mass quality. In most cases, scanline surveys from a series of blocks were used to determine rock mass quality. The resultant N.G.I. index was then applied to all of these blocks, resulting in identical Q values for all of the blocks in the area. This produced the uneven distribution of the values as applied to the histograms.

In many cases, either footwall or hanging wall access for a given series of blocks was not available. As such, the value of Q for the accessible wall was applied to the other wall, resulting in very similar histograms for both hanging and footwalls. In general, based on a quick visual inspection of the scanline survey data, the

hanging and footwalls did display very similar rock mass quality characteristics within each ore zone.

As no major support system such as backfill or pillars exist inside the VBM stopes during mining, a reasonable rock mass quality is required in the stope back and walls in order for the excavation to be self supporting. In general, the more competent the rock mass, the larger the excavations that can be created. Figures 2.33 and 2.34 illustrate the distribution of rock mass strength expressed in terms of the N.G.I. index for stope backs and walls in Canadian open stope mines [Potvin (11) (1988)]. Comparing the Thompson Mine rock mass quality with these indicates that the Thompson hanging walls are of slightly better quality than the canadian average, but the stope back quality is below the average. The similarity in Q indices for Thompson stope backs and walls indicates a consistency of joint characteristics throughout the rock mass.

2.6 Stress

Stresses which exist in an undisturbed rock mass are related to the weight of the overlying strata and the geological history of the rock mass. This stress field is disturbed by the creation of an underground excavation and, in some cases, this disturbance induces stresses which are high enough to exceed the strength of the rock. In these instances failure of the rock adjacent to the excavation can lead to instability which may take the form of gradual closure of the excavation, back falls, slabbing of side walls, or in extreme cases, rock bursts.

The excavation produces a redistribution of the stress field. This new induced stress state is three-dimensional and extremely variable in magnitude and direction and may reach a magnitude several times that of the virgin stress. The determination of the probable location and magnitude of stress concentrations is therefore of paramount importance in evaluating stability.

Thompson Mine and CANMET engineering personnel carried out in-situ stress measurements on 2000L and 4000L during 1978. The test results predicted the mean compressive stresses to be oriented as follows:

Maximum stress	NNW/SSE, dip 10 degree NNW
Intermediate stress	ENE/WSW, dip 18 degree ENE
Minimum stress	NE/SW, dip 70 degree SW

The magnitude of stresses measured at the Thompson Mine is given in Figure 2.35. The major and intermediate principal stresses ($\sigma 1$ and $\sigma 2$) are essentially horizontal while the minor principal stresses are near vertical and appears to parallel the foliation, as shown in Figure 2.36.

From an examination of regional geology maps, it is apparent that σ_1 follows the trend of the MacKenzie dyke swarm. These dykes were intruded along extensional features in the earth crust and represent the last major phase of tectonism in the region.

The stress directions obtained correlate well with geological observations and are in agreement with results from Canadian Shield as a whole, as illustrated in Figure 2.37.

One of the major objectives of a stress measurement program is to establish the average stress gradient. Hoek and Brown (8) (1980) have reviewed the results of stress measurements undertaken throughout the world. From these measurements, taken in a variety of geological environments, they made the following conclusions:

1. The vertical stress (σ_v) can be reasonably estimated from the weight of overlying rock (Figure 2.38)

2. The average horizontal stress (σ_h) in MPa generally falls within limits defined by:

where k is average horizontal to vertical stress ratio and Z is the depth in meters at which the stress is required (Figure 2.39).

Two graphs have been constructed from Thompson Mine stress measurement results. The first of these is a plot of the vertical stress, σ_v, calculated at each measurement site against depth (Figure 2.40). The second graph is a plot of the ratio of average horizontal stress to vertical stress (K) against depth (Figure 2.41).

Thompson Mine stress measurement results fall within the world wide data proposed by Hoek and Brown (8) (1980) and the extreme value gradient for Canadian Shield rocks.

By linear regression analysis it has been postulated that the vertical stress at Thompson Mine can be calculated by the equation:

$$\sigma_v = 0.83 + 0.032*Z$$

where σ_v = vertical stress in MPa
Z = depth in meters

A curved line has been fit to the data in Figure 2.41 to represent the gradient of the average ratio of horizontal stress to vertical stress.

In conclusion, this section of the report summarizes the determination of the regional in-situ stress field. Measurement of the localized induced stress in the underground stoping areas is detailed in the field instrumentation section of this chapter.

2.7 Visual Monitoring

2.7.1 Introduction

In carrying out a visual monitoring program, active underground areas should be visited on a regular basis to record relative changes in ground conditions. These changes can be identified by the degree of difficulty experienced in performing specific mining activities. A typical underground monitoring system could include section sketching, blasthole surveys and drawpoint monitoring. The monitoring will give a good indication of the change in conditions in a stoping area, and will be invaluable in subsequent back-analysis and calibration of numerical and empirical models.

As a part of this CANMET project, visual monitoring was attempted on a trial basis in some stoping areas at the Thompson Mine. The following indicates the procedures that were followed in implementing this program.

2.7.2 Section Sketching

Section sketching involves the recording of observations during regular stope visits. This procedure highlights the change in conditions with the advance of the mining horizon. Observations that were included at Thompson Mine are:

1) **Stress State:** The general conditions of the haulage ramp, drift or stope being inspected were described using the four-fold classification system outlined in Figure 2.42.

2) **Type of Failure:** The type of failure that occurred was recorded according to the nine simplified failure modes illustrated in Figure 2.43.

3) **Scaling:** The amount of effort required to scale the area was recorded using the three-fold classification described in Figure 2.44.

4) **Support Type:** A record of installed support was maintained using the support categories outlined in Figure 2.45. Where appropriate, the support dimensions were given (i.e. size and spacing of

bolts).

5) **Support Load:** The condition of the support system was monitored with respect to load. Figure 2.46 illustrates a classification that was used. When recording load condition, the percentage falling within each category was noted.

6) **Stope Stability:** A record was maintained of mining efficiencies and production statistics for each stope. This record provided information regarding tons blasted, ore recovered, dilution experienced, and volume of backfill placed.

A typical recording sheet for the section sketching operation is presented in Fig. 47.

In addition to systematic observation, the use of underground photography is highly recommended. Photographs of specific events provide a valuable permanent and visible record.

2.7.3 Blasthole Surveys

Blasthole performance is often a reliable indicator of changing conditions. Blastholes were therefore monitored on a regular basis. Two pieces of information were recorded, namely blasthole conditions and the number of redrilled holes in a specific area.

Blasthole conditions were recorded using the classification system shown in Figure 2.48, while redrills were categorized according to the following three-fold system:

Class 1: 0% to 10% redrills
Class 2: 10% to 50% redrills
Class 3: 50% to 100% redrills

A typical recording sheet is illustrated in Figure 2.49.

2.7.4 Drawpoint Monitoring

Systematic observation of the type, size and shape of the muck that appears at a drawpoint provides information on stope stability. Such a record also provides data with regards to the success of primary blasting procedures. A typical drawpoint recording sheet is presented in Figure 2.50.

2.7.5 Reporting of Results

The monitoring program as described above should be carried out on a regular basis. The time interval between inspections will depend on how quickly change in underground conditions are occurring. Initially all active areas should be inspected monthly. Where warranted (i.e. those areas with rapidly changing conditions), the interval between inspections should be reduced.

The inspections should be carried out by an identified person reporting to the Mine Superintendent. After each inspection a summary report should be prepared. To assist in reporting, a mine longitudinal showing relative ground conditions in a colour coded form should be maintained. This longitudinal should be updated after every inspection.

Appendix E details a case history of section sketching for the 1803-330 stope, and presents a plan displaying the results of a single visual monitoring excursion.

2.8 Field Instrumentation

2.8.1 Introduction

Underground instrumentation can be used to detect potential ground movement and stress concentrations in the rock surrounding a mine excavation. It is advantageous to be able to predict these occurrences as they can ultimately jeopardize the safety of the mining operation. The measurements can provide warnings of potential problems, and as a result preventive measures can be implemented before the problems have progressed to a stage whereby the remedial measures are either very expensive or impossible.

Two types of instrumentation will be described in this section, those being ground movement instrumentation and load stress measurement.

2.8.2 Ground Movement Instrumentation

The most common techniques of monitoring rock mass displacement are best summarized as follows:

1) **Surveying** - used to determine the displacement of targets fixed to the surfaces of underground excavations relative to a stable base station.

2) **Convergence** - measurements between targets attached to the wall and back of an excavation by means of a tape or extensometer in order to record the relative closure of an excavation.

3) **Borehole Extensometer** - used to measure displacement in the rock mass surrounding an excavation. It consists of either sliding rods or tensioned wires, anchored at selected points within the borehole. A measuring device physically measures the position of the anchor relative to the collar.

The second of these techniques, convergence measurement, is preferred over the other two. Reasons for this include the relatively low cost of a system, the ease of installation, and the reliability of the results.

2.8.3 Local Stress Measurement

The most popular type of stress measuring device is the borehole vibrating wire stress meter. It is practical, rugged and gives an indication of relative stress change from a single borehole. The instrument consists of a hollow steel cylinder containing a pre-tensioned steel wire oriented diametrically across the cylinder. The stress meter can be anchored in the borehole using epoxy resin or a wedging device. A portable readout unit is used to vibrate the steel wire and it's resonant frequency can be converted to the relative stress value along the axis of the wire.

With this device, stress change is measured at a small point within the rock mass and can be affected by factors such as:

1) The modulus of the individual rock type in which the stressmeter is set.

2) The pressure of jointing in the vicinity of the meter, which may give rise to high localized shear stresses.

3) If the rock mass is shedding load, individual block movement may occur. This will give rise to very large variations from one block to the next.

4) The cleanliness and smoothness of the hole surface will affect the intimacy of contact between the gauge and the borehole wall.

5) If the rock material is very weak and the gauge is very stiff, then the rock could fail around the gauge seating.

6) The wedge anchoring device could slip during blasting.

The vibrating wire stress gauge is not sensitive to electrical connections, heat, humidity or the creep of glues, and relies on a robust and positive mechanical setting. Therefore if the rock is solid, behaves elastically, and the rock type is known, only points 4 and 6 are critical.

2.8.4 Stress Measurement at Thompson Mine

At Thompson Mine, the majority of the currently installed instrumentation systems are located in the 1803-330 stoping area. This arrangement is designed to allow for collection of data on ground movement displacement and stress measurements in one specific area of the mine. Correlations between instrumentation data and visual monitoring data allow for interpretation and analysis as mining progresses in the area. Appendix F outlines the case history of field instrumentation in the 330 stope.

Field instrumentation programs are generally expensive, and most have associated problems and inaccuracies. The success of a program is extremely dependent on the number of installations successfully achieved, and the number of readings taken. The rate of success of individual installations is approximately 50% in the mining industry. For this reason, occasionally it becomes more economical to establish a field instrumentation program in one specific area of the mine and relate the results to other areas.

3.0 MINE OPERATIONAL DATA COLLECTION

3.1 Introduction

In identifying the individual steps followed to develop and mine a stope using vertical block mining techniques, the following events, listed in chronological order, represent the normal sequence:

1) Develop top and bottom sill horizons
2) Condition ground as required
3) Drill appropriate production hole pattern
4) Begin loading and blasting production holes
5) Begin muck removal from drawpoint on bottom horizon
6) Complete blasting and then muck out block completely
7) Fill block with cemented sandfill
8) Begin mining cycle on adjacent block

In longitudinal section, the standard arrangement illustrating various stages of this sequence can be viewed in Figure 2.51.

At Thompson mine, the terminology used dictates that a single section of ore to be mined in one cycle (as per the stages listed above) is called a block. A series of blocks, joined either longitudinally, transversely or vertically, is termed a stope (Figure 2.52). (Note: It is realized that this is in direct opposite to the adopted industry standard, however this can be explained as a result of the conversion at Thompson Mine from cut and fill to vertical block mining. The mine terminology has not, as in many other cases, been changed with the change in technology.)

Two fundamental arrangements of vertical block mining are used at Thompson Mine: longitudinal and transverse. Longitudinal VBM involves silling of the top and bottom portions of the ore zone of a given mining horizon along their entire strike length. Subsequently the blocks are mined in retreat order from one end of the stope to the other, as shown in Figure 2.53. This method is used primarily in the narrow sections of the orebody where an opening in the ore can be maintained for an extended period of time with minimal reconditioning of ground. This method is also used when economics dictate that extensive wall rock development in order to mine the block is not feasible.

The transverse vertical block mining involves extensive footwall (and/or hanging wall) rock development on both the top and bottom block horizons. The ore zone is accessed, one block at a time, from these

development headings. In general, the top and bottom sills in the ore zone are developed immediately prior to the mining of the block. This method is used in the wider areas of the orebody, and in areas where the sills developed in the ore cannot remain open for extended periods of time due to weak ground.

As in the case of vertical block mining terminology, many of the standard details associated with VBM design are fixed as a result of the conversion from cut and fill mining to VBM at Thompson Mine. An example of this is the location of the blocks in the mine. Figure 2.54 represents a histogram indicating the relative depths of the VBM blocks (from surface) at Thompson Mine. Most of the blocks are situated between 1000 and 2400 level at the mine. The distribution of the blocks is considerably varied. If a mine was developed solely for vertical block mining, the grouping on the same histogram would be much tighter. With time and additional mining, it is expected that the histogram in Figure 2.54 will identify a greater number of blocks in the lower reaches of the mine between 2000 and 4000 level.

The mine operating procedure database has been grouped into six categories for the purpose of presenting the results. Those categories include:

1) block geometry
2) stope development
3) production drilling
4) production blasting
5) production mucking
6) mining efficiency

Each of these activities will be covered individually in the following sections of the report. Although in some cases the results presented may not seem clearly logical, it must be remembered that the vertical block mining areas at Thompson Mine are conversions from old cut and fill workings. As such many parameters are not easily controllable, as in the case of sub-level interval spacing.

3.2 Stope Geometry

The Thompson Mine orebody is generally regarded as a steeply dipping orebody. The average dip of the orebody is 69 degrees, as illustrated in Figure 2.6. The range of dips varies from 45 degrees to 90 degrees in the vertical block mining areas of the mine.

The stope block dimensions have in general been limited in an attempt to reduce the amount of wall dilution inherent to open stoping, particularly to VCR. In referring to Figure 2.55, the average height of a VBM block at Thompson Mine is 40 m (131 ft). The height can vary depending on factors such as the dip, the location of existing levels, the number of sublevels developed, and the relative elevation of the cut and fill stope prior to conversion. Where the geological outlines are inconsistent or indicate a narrow ore zone, block heights are reduced in order to initiate selective mining.

The width of the ore zone dictates the block width in most cases at Thompson Mine. In only two cases was transverse block extraction practiced, those cases resulting in high fill dilution rates and a massive back failure. In extremely narrow zones (i.e. less than 3 m or 10 ft wide), the block width is usually greater than the ore zone width, and subsequently waste rock is blasted intentionally as part of the block. In referring to Figure 2.56, the average block width of the 126 VBM blocks selected in the database was 8.8 m (29 ft).

VBM block lengths are generally designed with the width and height dimensions taken into consideration. This dimension may be decreased in anticipation of poor hanging wall conditions. Conversely, the length can be increased to reflect production constraints. As a series of blocks are mined within a given ore zone, experience will dictate adjustments to the block length in order to achieve an optimum length without experiencing severe hanging wall problems. In referring to Figure 2.57 the average VBM block length at Thompson Mine is 17.4 m (57 ft).

3.3 Stope Development

As outlined in the introduction to this section, two types of vertical block mining are utilized at Thompson Mine: longitudinal retreat and transverse. The development scenarios for each of these are considerably different.

Beginning with longitudinal retreat, the top and bottom mining horizons are usually silled along their entire strike length prior to any block being extracted. The top sill is generally developed with 3.7 m (12 ft) of headroom and at a width ranging from 3.7 to 6.1 m (12 to 20 ft). Ground conditioning in the top sill occasionally includes 6 m (20 ft) cable bolts, which are installed in order to stabilize the back once a block has been extracted beneath. The bottom sill is generally opened to the full width of the ore zone, particularly as retreat stopes are typically located in narrow vein areas. Some extra ground support (resin rebar or cable bolts) is usually installed in the back where future mucking brows are to be located. In most cases where a stope is in direct conversion from cut and fill to VBM, the bottom sill is already established and requires only some additional ground support.

The development sequence for transverse VBM follows a different set of procedures. A large portion of the rock development is performed prior to any block mining. Once a block is ready to be developed, the top and bottom sills are excavated. As transverse stoping is typically employed in wider ore zones, neither the top nor bottom horizons are silled out to the extent of the block. The silling may stop short of extending from hanging wall to footwall contact, or it may not project the along the entire strike length of the block.

The typical drawpoint spacings used for transverse block mining at Thompson Mine are represented in Figure 2.58. From this histogram, it is apparent that the drawpoint spacings from centre to centre range from 7.6 to 18.3 m (25 to 60 ft), with 12.2 m (40 ft) being the average spacing. The length of the block along strike is usually a determining factor in determining these distances.

Figure 2.59 depicts a histogram illustrating the variations in the width of pillar established between the footwall (or in some cases the hanging wall) extraction drift and the ore zone for transverse blocks. These pillar widths range from 9.1 to 24.4 m (30 to 80 ft), with the average being approximately 15.2 m (50 ft). In general, the pillar width is a function of the depth of the stope in the mine - the deeper the stope, the higher the virgin and induced stress field, and subsequently the greater the design pillar width.

The specific ore zone in which the development is to be located also affects the design width of this pillar. In the 2 shear ore zone, for example, footwall access drifts are maintained an average of 24.2 m (80 ft) away from the orebody due to the poor ground and high induced stress conditions common to the 2 shear area. It should be noted that as the degree of remnant mining at Thompson Mine increases, stress levels are also expected to escalate, and subsequently haulage drifts will be located even further away from the ore zone.

3.4 Production Drilling

A combination of both 115 mm (4.5 in) inch and 165 mm (6.5 in) in-the-hole drills are used for production drilling at Thompson Mine. During the initial stages of conversion to vertical block mining at the mine, 165 mm (6.5 in) drills were predominantly used, those being the industry standard in the early 1980's. When 115 mm (4.5 in) drills were later introduced, logistics determined that the 115 mm (4.5 in) drills would be used primarily in the narrow ore zones, with the 165 mm (6.5 in) drills reserved for the stopes in the wider ore zones.

One method used at Thompson Mine to evaluate drill efficiencies is through determining the total feet drilled per shift. Figures 2.60 and 2.61 represent the footage drilled per shift for 115 mm (4.5 in) and 165 mm (6.5 in) drills respectively. The respective averages of feet per shift are 35 m (115 ft) and 32 m (105 ft), a working shift being defined by an 8 hour period. The differences between the two sizes of drills measured on this basis are minimal. These factors can vary dramatically from one stope to the next depending on the type of drill, the number of hole set-ups, the length of hole, the ore and rock type and quality, the number of miners per ITH drill crew, operator experience, and equipment down-time for maintenance and repairs.

Another method used to evaluate drilling efficiency is the calculation of the factor representing the tons of ore broken per foot of hole drilled for each block. From the histograms in Figures 2.62 and 2.63, the average value of tonnes per meter (tons per ft) drilled for blocks drilled with 115 mm (4.5 in) and 165 mm (6.5 in) holes are 7.6 (2.6) and 13.2 (4.5) respectively. This represents a 73% increase in tonnage per meter (foot) drilled for a 108% increase in cross-sectional area of the hole, which appears to be within reason.

Hole burden and spacing, ore zone width, and the number of holes requiring redrilling due to stress related problems or hole deviation can influence this factor. In addition, the location of the drill horizon for some blocks is in the hanging wall, which results in fanning of holes to intersect the hanging wall and footwall contacts. In this case, the calculation of the tons per foot drilled factor is artificially low, as the top segment of each hole is not used for blasting purposes.

In summary, this section presents a brief overview of the relative sizes of production drills used at Thompson Mine, and examples of how data collection is used to monitor drilling and mining efficiencies. Chapter 4 of this report, Drilling and Blasting, describes the drilling practices at Thompson Mine in far greater detail.

3.5 Production Blasting

The primary method of production blasting used at Thompson Mine is cratering for both 115 mm (4.5 in) and 165 mm (6.5 in) holes. Amex (ANFO) is used primarily in dry holes and Aquamex in wet holes. In the initial stages of VBM mining, the standard charge weight for a 115 mm (4.5 in) blasthole was 12.5 kg (27 lbs) for both Amex and Aquamex and 25 kg (55 lbs) for 165 mm (6.5 in) blastholes. These charge weights were used for years as they were the standard explosive packaging weights as provided by the explosive manufacturers. As a result, the size of the Amex explosive charge related more to a cylindrical charge theory, while the Aquamex charge related more to the spherical charge theory. Recently, the Amex charge size has been changed to 18.2 kg (40 lbs) for a 165 mm (6.5 in) hole in order to reflect the spherical charge theory in maintaining the explosive charge length to within six times the diameter. This change in charge weight has proven to crater the ore with a minimal reduction in break per charged hole. Also, production holes located along the footwall and hanging wall are occasionally loaded with lighter charges in order to reduce wall damage.

Secondary blasting of large boulders in the drawpoints is not considered in the calculation of a powder factor for blocks at Thompson Mine. This is due to the unavailability of data relating to the degree of secondary blasting.

The distance from the breakthrough of the blasthole to the centre of the explosive charge is referred to as the standoff. The current standard standoff was established through experience at Thompson Mine. The average standoffs for 115 mm (4.5 in) and 165 mm (6.5 in) holes are 1.2 m (4 ft) and 1.8 m (6 ft) respectively. Standoff distances are occasionally reduced if blastholes are found to be chambering, which is indicative of highly stressed ground or unusually strong rock. This reduction in standoff is rarely successful - more often extra holes are drilled into the pattern in order to cure the chambering problem.

One common method used to monitor blasting efficiency is through the use of powder factors, which at Thompson is related in terms of the total explosive weight divided by the tonnage of muck broken. Figures 2.64 and 2.65 illustrate the variation in powder factors at Thompson Mine for both 115 mm (4.5 in) and 165 mm (6.5 in) holes. The average powder factor is 0.5 kg/tonne (1.0 lbs/ton) for both sizes of production holes. Equal powder factors such as this should be expected if blasts designs are proper. Otherwise, a significant difference in powder factors between the two hole sizes would indicate that over-charging in one size of hole was occurring.

Powder factors can vary based on a very large number of variables. Some of these include ground strength and quality, type of blasting (i.e. crater versus slash), stress regime, hole pattern, type of explosive, method of detonation, degree of cross-initiation, degree of overcharging, type of stemming and so forth.

Detailed descriptions of blasting methodology at Thompson Mine are discussed in greater detail in

3.6 Production Mucking

The Thompson Mine, originally being equipped for cut and fill mining, entered the mining method conversion stage with only 1.5 cubic m (2 cubic yd) scooptrams. As the demand for more or newer scooptrams has transpired, the new units have been of either 2.7, 3.8, or 4.6 cubic m (3.5, 5, or 6 cubic yd) capacity. Most new 4.6 cubic m (6 cubic yd) machines are outfitted with only 3.8 cubic m (5 cubic yd) buckets due to the heavy specific gravity of the Thompson ore. In addition, all scooptrams designated for VBM mucking purposes are equipped with remote control systems in order to allow safe ore removal from the open blocks.

The histogram in Figure 2.66 represents the average tonnage mucked per shift from the Thompson Mine VBM blocks. This represents the tonnage hauled from the block drawpoint to the stope ore pass in an average 8 hour shift. From the histogram, the average tonnage indicated is 427 tonnes (470 tons). It is fully expected that this average will increase in future years as larger equipment continues to replace the obsolete 1.5 cubic m (2 cubic yd) scooptrams. The tonnage mucked per shift is dependant on a number of variables, the key ones being the scooptram size, the haul distance, the scooptram availability, and the fragmentation of the muckpile.

In assessing the average haulage distances common to VBM mucking operations at Thompson Mine, Figure 2.67 presents a histogram of these distances. The average distance indicated is approximately 105 m (344 ft).

In many of the stopes that were converted from cut and fill to VBM, a number of ore passes are located throughout the length of the stope. The option therefore exists to dump into more than one orepass, particularly if problems are being experienced in the orepass closest to the brow (i.e. full pass, hangups, etc.) Ideally if an orebody was being developed for VBM mining as opposed to being converted, the maximum design haulage distance would be approximately 152 m (500 ft) in order to maximize efficiency. However at Thompson Mine, this optimum distance is frequently exceeded due to the fixed location of these passes as a result of previous cut and fill operations.

Details of production mucking at Thompson Mine will be covered in greater detail in Chapter 5, Stoping Procedures.

3.7 Mining Efficiency

Four parameters that can be used to evaluate the relative efficiency of a particular mining operation or mining method are:

- dilution
- recovery
- productivity
- cost

Each of these items will be discussed in some detail with respect to vertical block mining at Thompson Mine in this section of the report.

3.7.1 Dilution

Dilution represents one of the most important efficiency factors on the list. The ranges of the other three factors are limited within a given mining method. Dilution, however, can vary considerably, particularly in the case of VBM mining.

Two primary sources of dilution are associated with VBM mining at Thompson Mine. These are dilution from the wall rock and dilution from the sandfill in adjacent mined out blocks.

Neither form of dilution is easily measurable. Wall rock dilution originates from the hanging and/or footwalls of the VBM blocks. It can usually be visually identified in the drawpoint as the muck is drawn. However, the Thompson Mine ore characteristically contains frequently occurring rock inclusions which are not considered as dilution. Differentiation of these inclusions from wall rock dilution in the drawpoint is extremely difficult, if not impossible. Surveying or visual observation of the stope walls after a block has been emptied is also unrealistic. This is particularly true with longitudinal retreat mining, as only a portion of the hanging wall can be observed once the block is open. Standard survey equipment is not capable of measuring these over-breaks on the walls, as physical access to the wall is necessary for measurement of this distance with standard survey equipment. Calculation of wall dilution in comparing the calculated block tonnage versus the actual tonnage removed is also subject to inaccuracies. The calculated block tonnage is at best ±10% due to the undulating nature of the hanging wall and footwall contacts. The figure representing the tonnage removed is at best ±20% due to inaccuracies with the drawpoint count and fluctuating bucket factors.

In summary, dilution from wall rock at Thompson Mine is not a measurable entity. From experience, it has been assumed that inherent wall rock dilution in the VBM stopes is between 10% and 15%. Rock dilution due to wall instability and sloughing is over and above this 15% dilution figure.

Dilution from sandfill is equally, if not more difficult, to measure. This dilution can originate either from the filled blocks adjacent to the open stope, or from the mucking floor which is usually the last filled cut of an old cut and fill stope, or the top of a previously mined VBM block. A major problem with quantifying sandfill dilution is the inability to visually identify the sandfill in the drawpoint. A minimum of 10% sand must be present in the muck pile before it can be visually identified. Once it is observed, an estimate is made of how much sand is physically in the ore (% by weight). This figure represents only an eyeball estimate and can potentially be in error by several orders of magnitude. Visual observation of the final stope opening is also of no practical value in determining the sandfill dilution, for the same reasons as for wall rock dilutions.

Through experience, a figure of 10% inherent sand dilution has been assumed for vertical block mining at Thompson Mine. This figure is applied to stopes where one of the four vertical walls of the open stope is a self supporting wall of sandfill. In the case of stopes that have two or more sand walls, the sand dilution would potentially be much higher.

3.7.2 Recovery

Recovery is another indirect measure of the efficiency of a mining operation. The mining method(s) being used is the most important factor influencing recovery.

Theoretically, vertical block mining is designed to achieve 100% recovery under ideal conditions. No pillars are designed to be left between blocks, and the open stopes are intended to be mucked until they are empty. The crown pillar left above the top sill is generally recovered using underhand or bulk pillar recovery techniques.

Surprisingly enough, in most cases 100% recovery is achieved at Thompson Mine. Mucking is terminated only if sufficient wall rock or sandfill has diluted the ore to a point whereby it becomes uneconomic to mine. This is a rare occurrence. In some stopes, low grade zones are left as pillars between blocks, but in general, pillars are not established between adjacent blocks. In general, overall recovery using vertical block mining is in the order of 95%.

3.7.3 Productivity

Productivity is another key measure of the efficiency of an operation. As vertical block mining is cyclical and dependant on tramming equipment sizes and haulage distances, average mine productivity in terms of tons per day per stope is difficult to quantify with any degree of accuracy. Productivity from any given stope can range from 180 tonnes (200 tons) per day to 1800 tonnes (2000 tons) per day.

One of the best measures of productivity is the ratio of tons per manshift. Hypothetically, a mine with

similar block sizes to Thompson Mine using 100% vertical block mining would reach a maximum productivity of approximately 18 tonnes (20 tons) per manshift. Productivity currently is 13.88 tonnes (15.27 tons) per manshift at Thompson Mine. One reason for this difference in ideal versus actual production levels is that other mining methods (i.e. underhand and cut and fill) are currently in use at Thompson Mine. Also, the mine remains in a late conversion stage with considerable rock development being performed. Productivity for Thompson Mine when it was a cut and fill operation was 25 tonnes (28 tons) per manshift.

3.7.4 Cost Efficiency

One of the most widely used indicators of efficiency, particularly in the comparison of mining methods, is the ratio of cost per unit tonnage. Vertical block mining at Thompson Mine is currently averaging $18.23 per tonne ($16.57 per ton). This figure represents only direct costs (i.e. labour, materials, repairs) and does not include the cost of development of the top and bottom sills.

Costs can vary substantially between vertical block mining areas. A block that is in a wide ore zone and has a short haul distance from the drawpoint to the ore pass can have direct costs of as low as $3.64 per tonne ($4.00 per ton). On the other hand, a small block using a 1.5 cubic m (2 cubic yd) scooptram to haul over a long distance can effect costs of as high as $18.00 per tonne ($20.00 per ton).

Another variation of the measure of cost efficiency used at Thompson Mine is the ratio of cost per unit weight product (cost per kg (lb) nickel at Thompson). This number, which is current $0.36 per kg ($0.80 per lb) at Thompson (mining costs only), is used more as an indicator of the overall mine efficiency. It can also be used to determine if a particular stope is economic to mine. Changes in this number are affected as much by a change in grade as a change in actual mining practice. As such, it is not used on a regular basis in comparing one stope to another.

It must be realized that all four of these indicators of efficiency (dilution, recovery, productivity and cost) can be interrelated. For example, dilution and recovery are directly related - an increase in recovery will most likely result in either an increase in dilution or an increase in cost. If recovery is reduced by eliminating small or low grade pockets of ore from the mining plan, then productivity will be increased and the cost per unit ton will decrease.

All four of these variables should be monitored individually as a mining operation progresses in order to measure and assess the overall mine efficiency.

4.0 SUMMARY

The development and maintenance of a well structured data collection system is of fundamental importance to a mining operation, regardless of the mining method being employed. At Thompson Mine, predominantly a vertical crater retreat operation, a data collection system was developed based on the case histories from the mining of 126 vertical crater retreat blocks.

The database was separated into two sections: geotechnical data collection and mine operational data collection. The geotechnical database included categorizations such as geology, structure, strength, and rock mass quality. The concept of stress and the effect of the redistribution of stresses through the use of visual monitoring and field instrumentation techniques was included in this geotechnical database.

The mine operational database was separated into categories such as stope geometry, development, drilling, blasting, and mucking. The mining efficiency, measured from the combination of parameters from these groupings, was subsequently derived from the mine operational database.

One important use of the database can be identified in the analysis of specific parameters from the database. A distribution of the typical mining block sizes, for example, is a simple yet useful comparison that can be produced from the database in a very short period of time. Based on the 126 block histories, the average block size at Thompson Mine was 40 m (131 ft) in height, 8.8 m (29 ft) in width, and 17.4 m (57 ft) along strike. This may appear to be a relatively obvious set of dimensions that could be reproduced without

the aid of a massive database, however in questioning various engineering and operational personnel regarding the average stope dimension, the number of different answers would vary linearly with the number of individuals asked.

Comparisons that are more complex than the one presented above are just as easily available from the data collection system. For example, the powder factors for production blasting using both 115 mm (4.5 in) and 165 mm (6.5 in) holes can be extracted from the database. In both cases, the averages prove to be identical to 2 decimal places (0.51 kg/tonne or 1.02 lb/ton). In absolute theoretical terms, this would be expected. However, in a practical sense, this relationship would not realistically be anticipated, particularly the accuracy to the second decimal place. What the comparison of the distribution of data from the two sizes of hole may indicate, though, is that through trial and error and with time, the charge weights and standoff distances have been adjusted in the field to a point whereby optimum breakage is achieved. This is not to suggest that the powder factors could not be significantly reduced in both cases, particularly given the inefficiencies of crater blasting within a fan pattern of holes. The data does indicate, however, that given the inherent blasting inefficiencies at Thompson Mine, production blasting is in a state of control.

The physical collection of data can also lead to discoveries of potential problems in the mine environment. For example, collection of geotechnical data in drifts and sills can identify poor ground conditions that can be accounted for in the design of a mining block. Also, visual monitoring can provide an indication of impending problems in current mining areas. An excellent example of this is documented in Appendix E.

The collection of data can also lead to establishment of levels of control and goals with respect to specific aspects of mine performance. For example, a database will be able to produce a set of representative data with respect to wall dilution. From this distribution, a goal or target can be developed with respect to the upper level of "acceptable" dilution within a given mining block. If this upper level is approached or reached, then a change in mining plan or stope design may be necessitated.

In conclusion, the effective utilization of a data collection system can be summarized best by the following list of activities:

- comparison of individual mining and geotechnical parameters
- analysis of operating parameters in order to determine levels of control
- indication of potential problems in current mining areas
- development of empirical relationships for the purpose of future mine design

The cost of establishing such a system may be relatively significant, however the cost of maintaining it will be minimal given the potential return from the database. As such, the development and maintenance of a data collection system is of paramount importance to the overall success of a mining operation.

5.0 REFERENCES

1. Barton, N., Lien, R., Lunde, J. (1974) Classification of Rock Masses for Design of Tunnel Support, Rock Mechanics, Vol. 6, No. 4.

2. Barton, N.R., and Bandis, S. (1982); Effect of Block Size on the Shear Behaviour of Joint Rock, Proc. 23rd U.S. Symposium Rock Mech. publ. AIME.

3. Bieniawski, Z.T. (1973) Engineering Classification of Jointed Rock Masses, Trans. S. Afr. Instn. Civ. Engrs., Vol. 15, pp. 335-344.

4. Bieniawski, Z.T. (1976) Rock Mass Classifications in Rock Engineering, Proc. Symposium on Exploration for Rock Engineering, Johannesburg, Vol. 1, AA Balkema.

5. Brady, B.G.H. and Brown, E.T. (1985) Rock Mechanics for Underground Mining, George Allan & Unwin, London.

6. Greer, G.J. (1989) Empirical Modelling of Open Stope Stability in a VCR Application at Inco's Thompson Mine, Inco Limited, Manitoba Division, Prepared for 91st A.G.M. of the C.I.M.. Quebec, Quebec.

7. Greer, G.J., Maynard, B.C., Platford, R.E., Sagman, J.E., Sutherland, I.M., Gerlinsky, B.F., (1986) VBM Stope Design Guidelines - 1403-301 Stope Stability Analysis, Unpublished report, Inco Limited, Manitoba Division, Thompson, Manitoba.

8. Hoek, E. and Brown, E.T. (1980) Underground Excavations in Rock, Institute of Mining and Metallurgy, London.

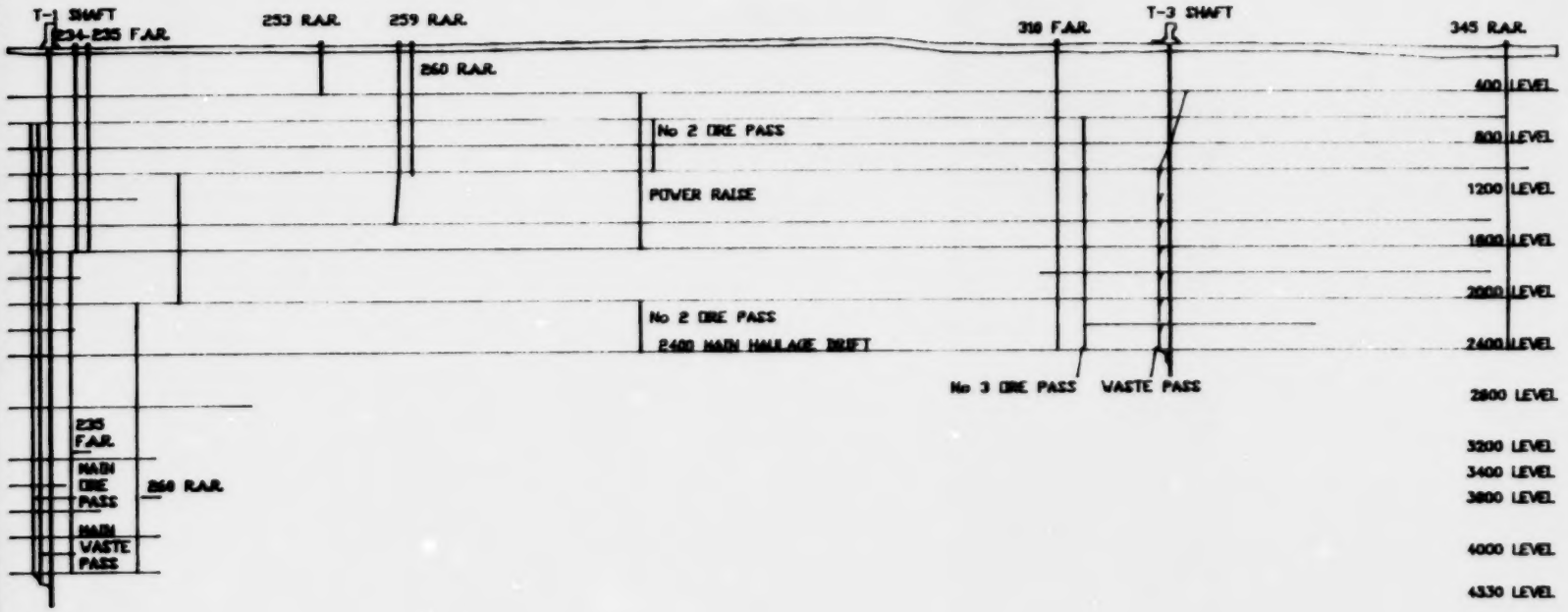
9. Mathews, K.E., Hoek, E., Wyllie, D.C., Stewart, S.B.V., (1981) Prediction of Stable Excavation Spans for Mining at Depths Below 1000 meters in Hard Rock, Canmet Report No. 17SQ 23440-0-9020.

10. Platford, R.E. (1989) Development of a Vertical Crater Retreat Mine Database at Inco's Thompson Mine, Inco Limited, Manitoba Division, Prepared for 91st A.G.M. of the C.I.M., Quebec, Quebec.

11. Potvin, Y., Hudyma, M., Miller, H.B.S. (1988) Integrated Mine Design, Volumes I-II, Prepared under NSERC Project No. CRD-8612, University of British Columbia, Vancouver, B.C..

12. Sagman, J.E. and Greer, G.J. (1986) Stope Stability Analysis - A Case Study, Inco Limited, Manitoba Division, Prepared for C.I.M. District 4 Annual Meeting, Winnipeg, Manitoba.

13. Steffen, Robertson & Kirsten (B.C.) Inc. (1984) Rock Mechanics Study - Thompson Open Pit, Appendices A-H, Vancouver, B.C..



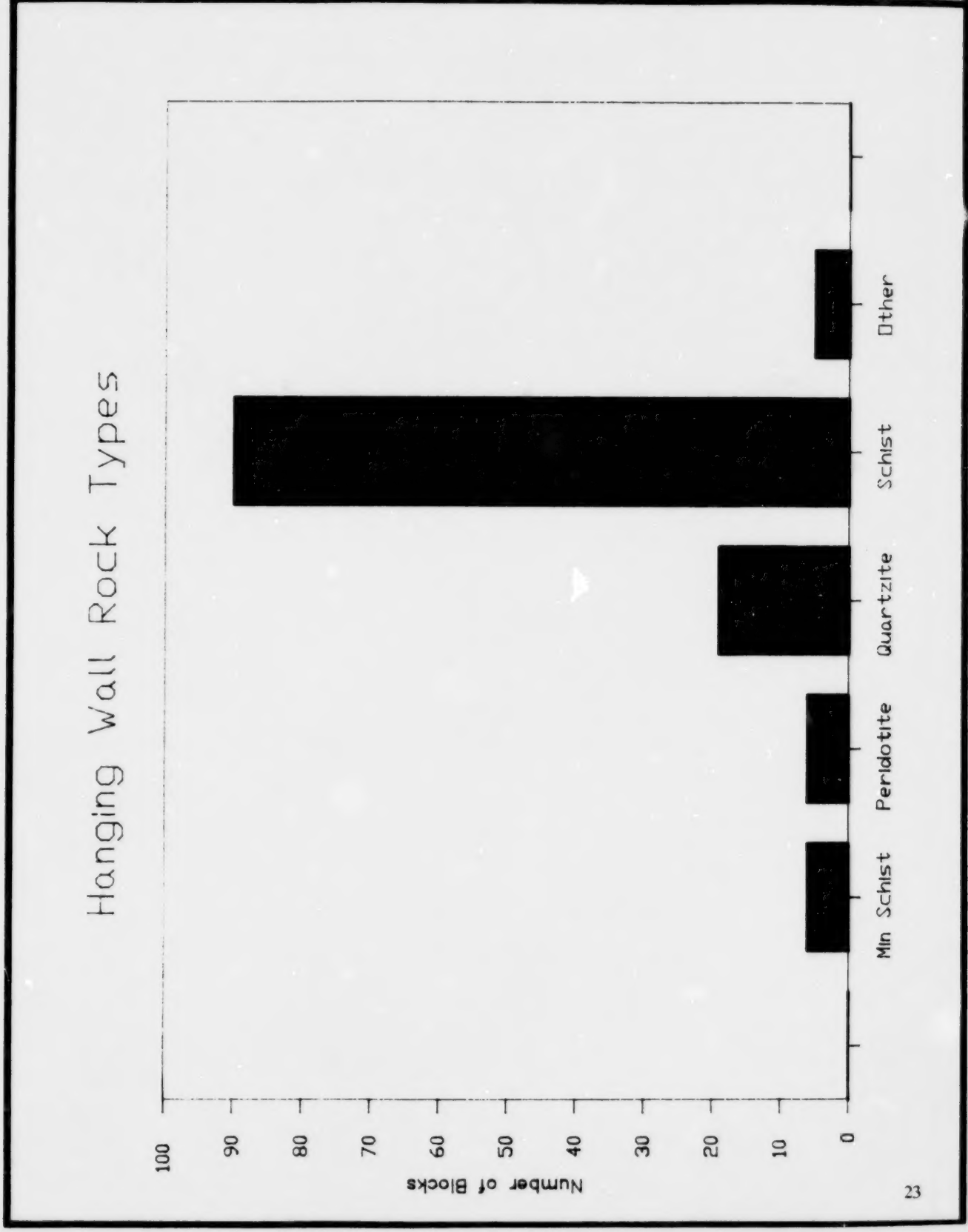
THOMPSON MINE LONGITUDINAL

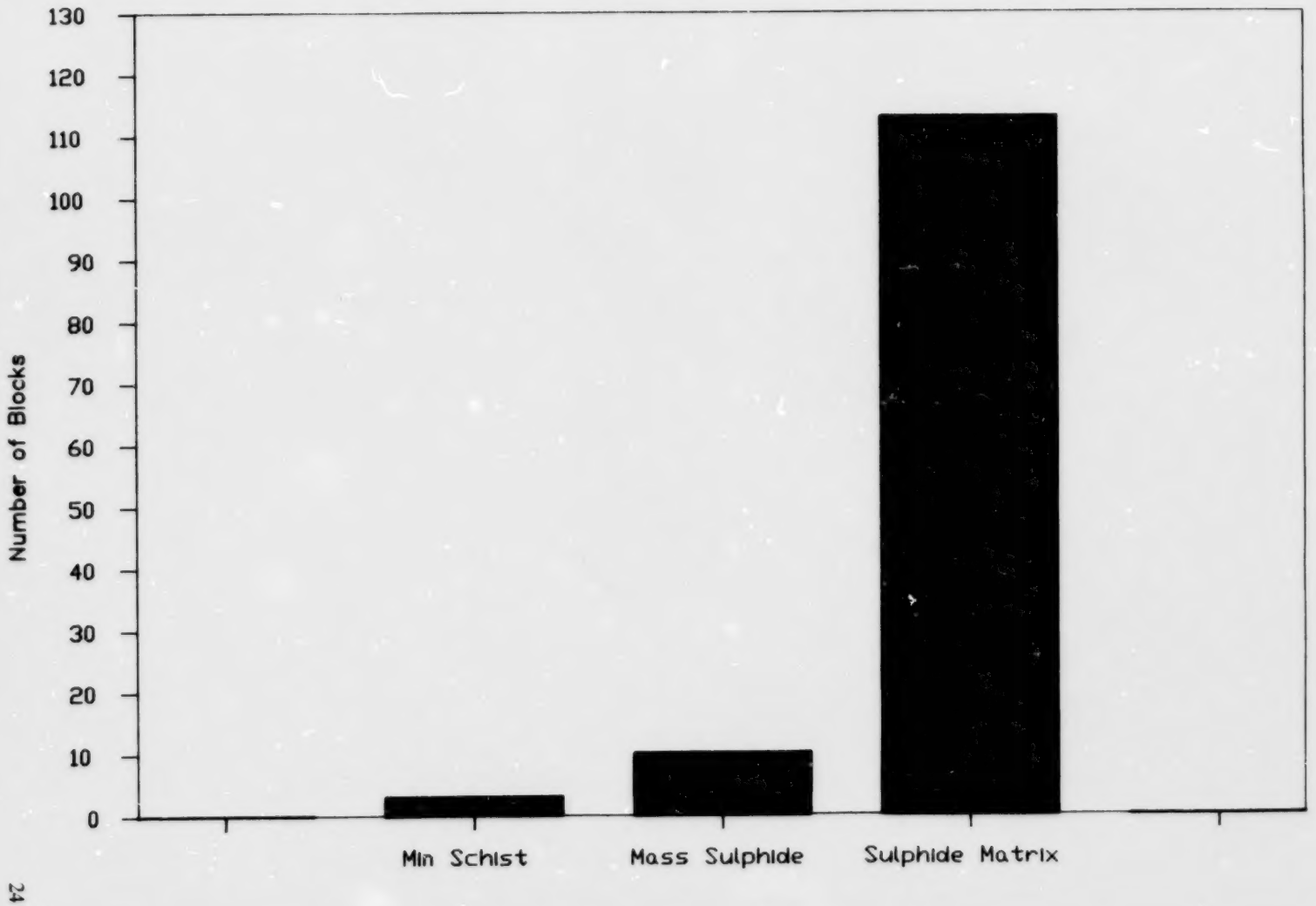
THOMPSON MINE LONGITUDINAL

FIG. 2.1

HANGING WALL ROCK TYPES

FIG. 2.2





ORE ZONE ROCK TYPES

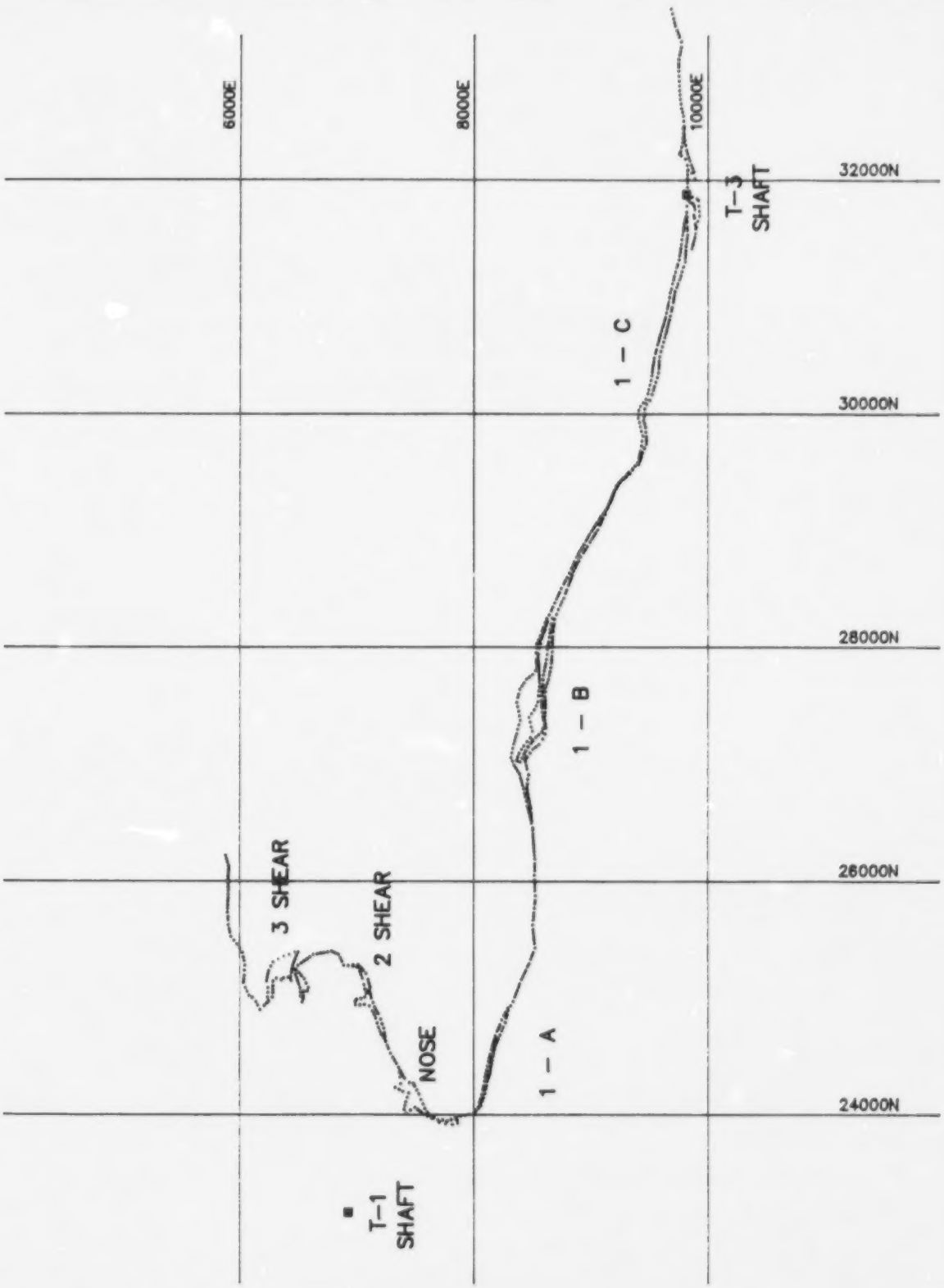
FIG. 2.3

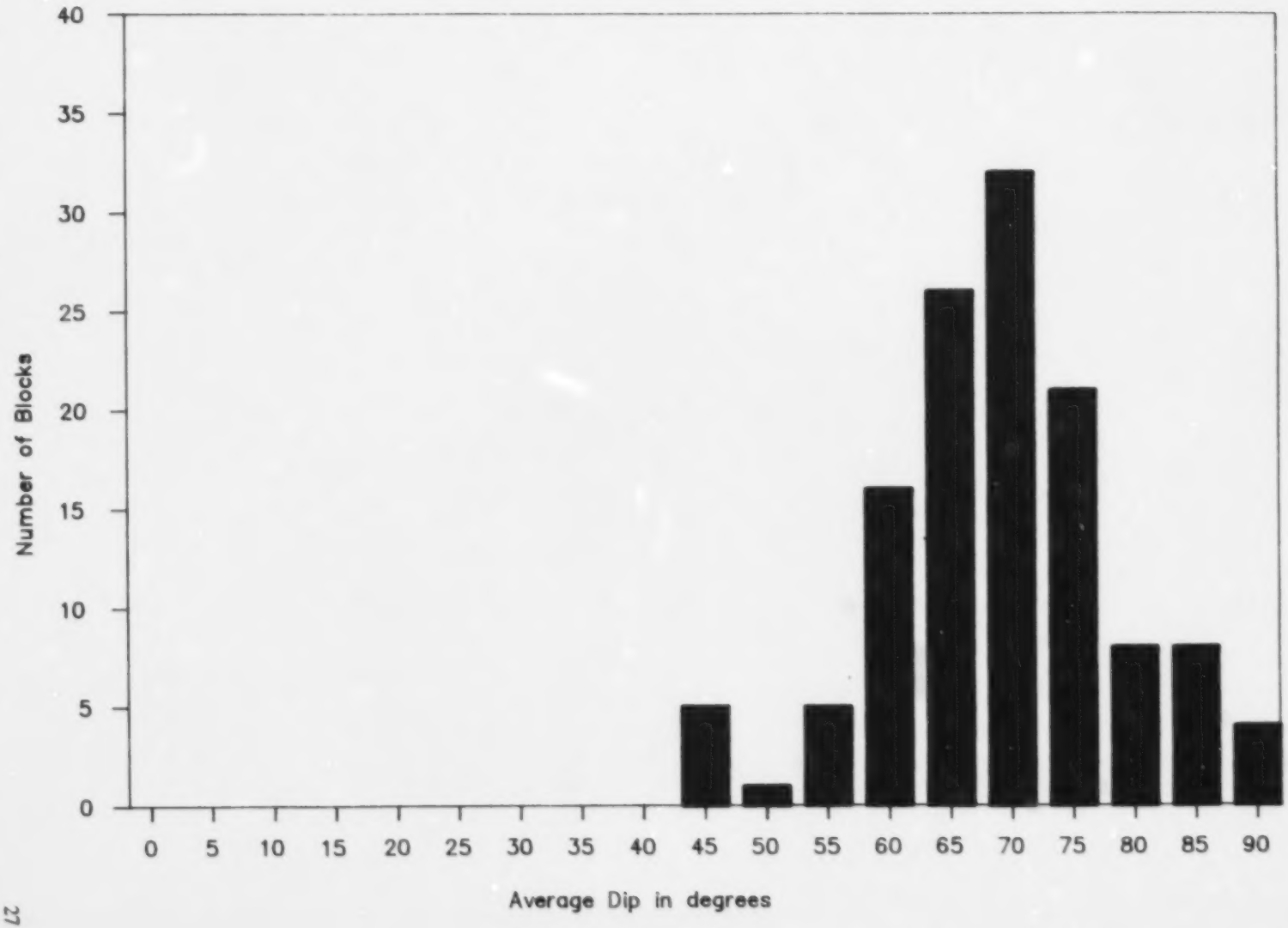
FOOT WALL ROCK TYPES

FIG. 2.4

PLAN OF REGIONAL STRUCTURE AT THOMPSON MINE

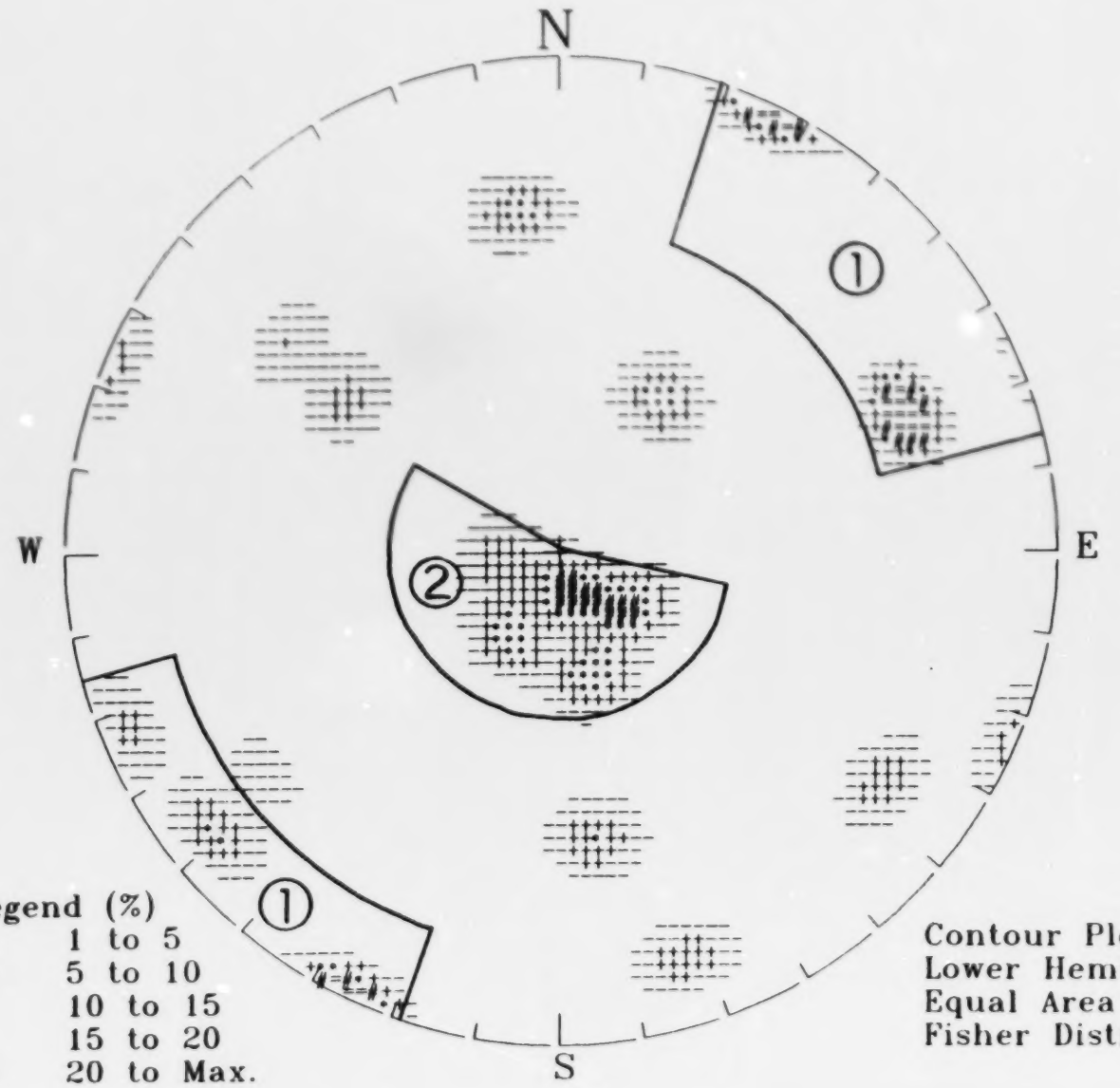
FIG. 2.5





DIP OF VBM BLOCKS

FIG. 2.6



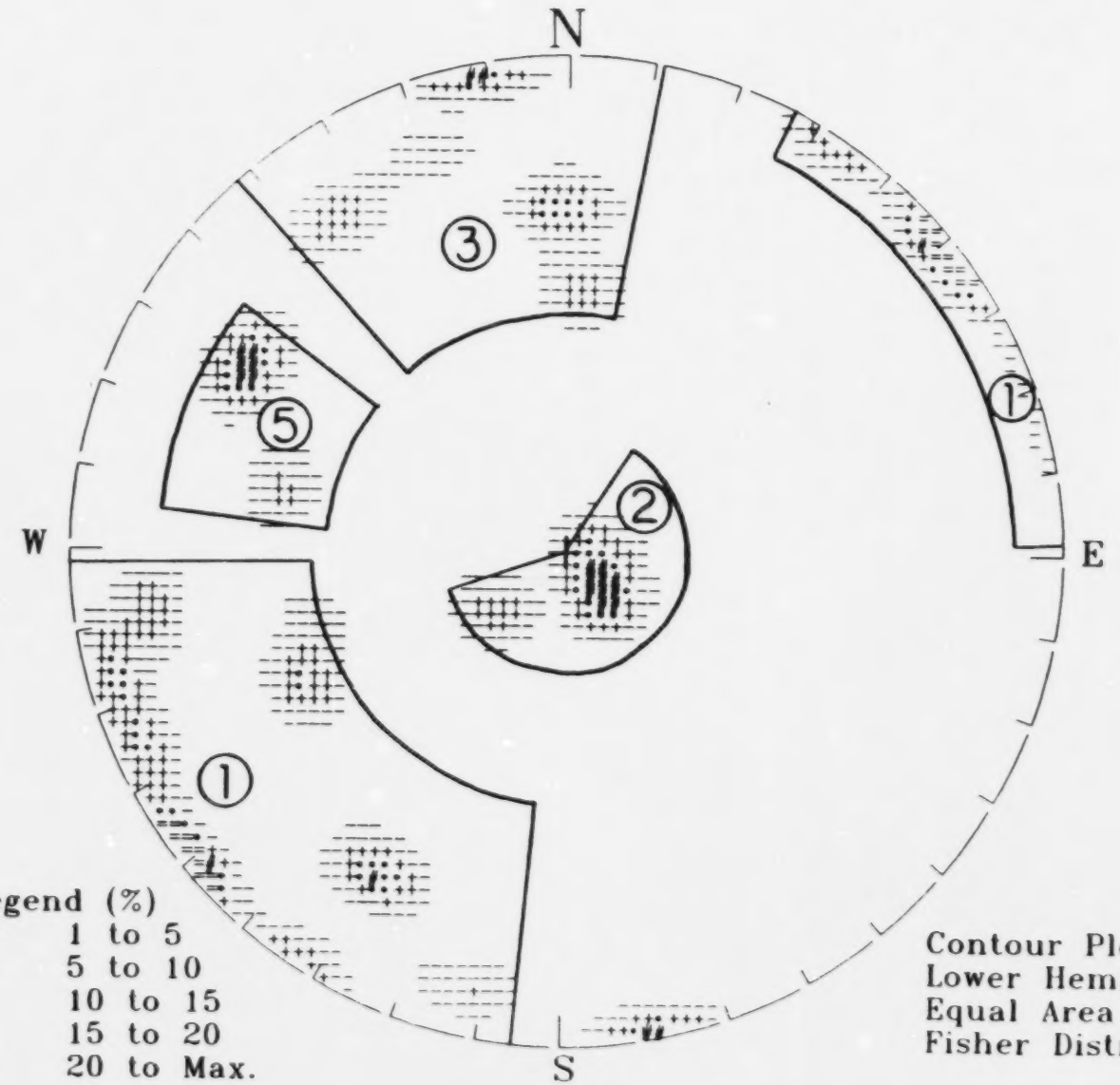
Legend (%)

----	1 to 5
++++	5 to 10
****	10 to 15
####	15 to 20
====	20 to Max.

Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

STEREONET – NOSE ZONE

FIG. 2.7



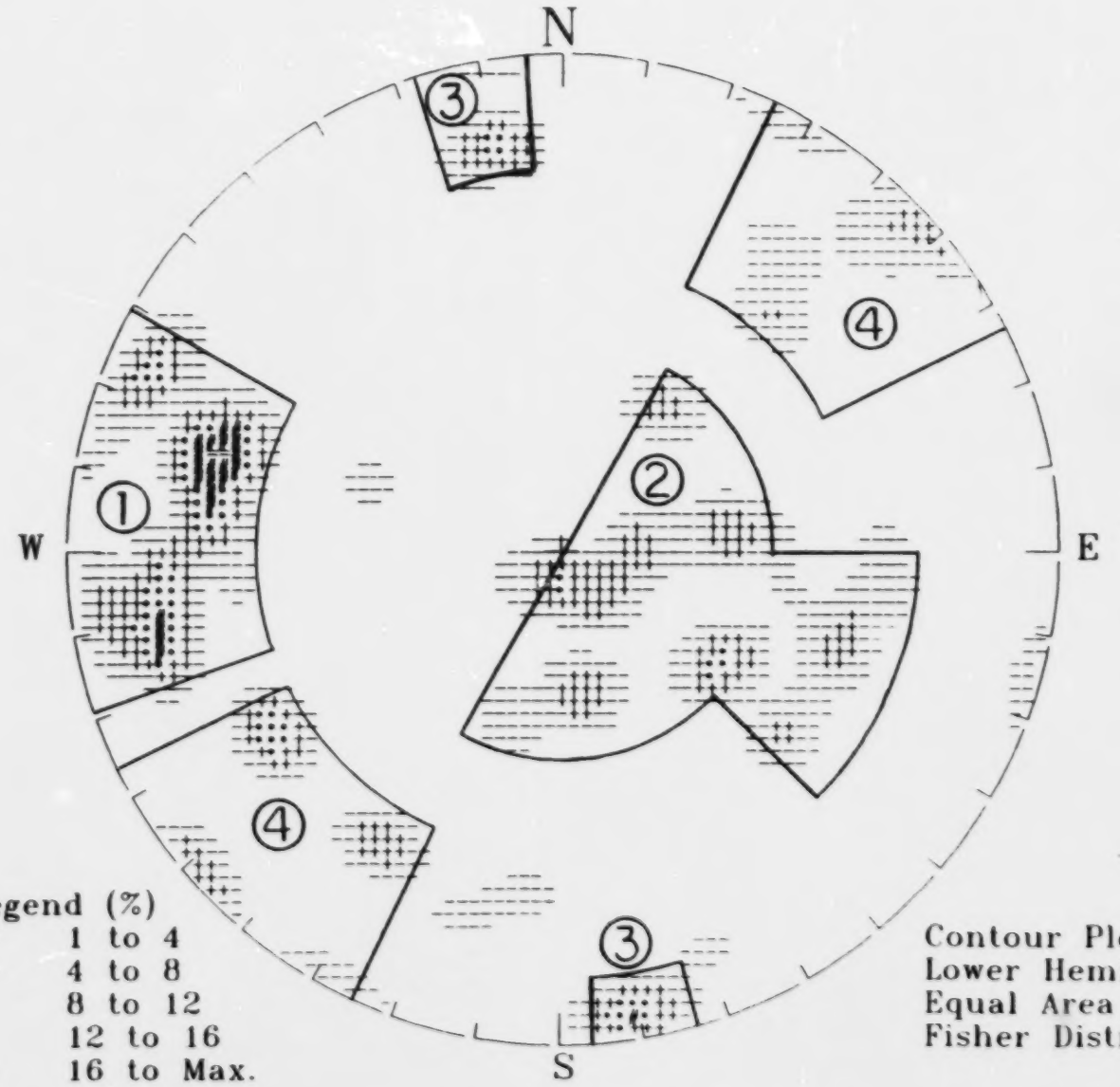
Legend (%)

----	1 to 5
++++	5 to 10
****	10 to 15
####	15 to 20
====	20 to Max.

Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

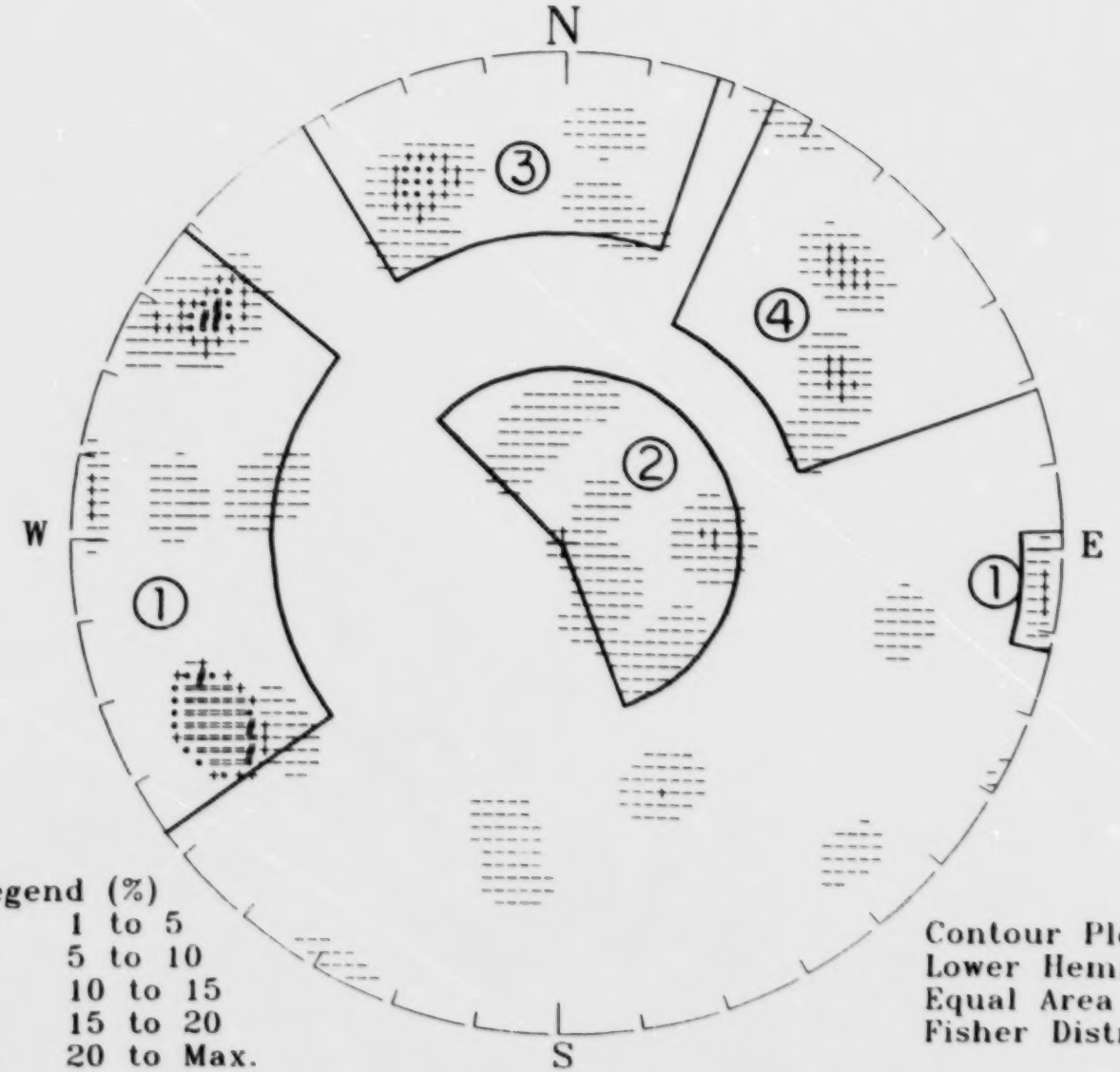
STEREONET – 2 SHEAR

FIG. 2.8



STEREONET – 3 SHEAR

FIG. 2.9



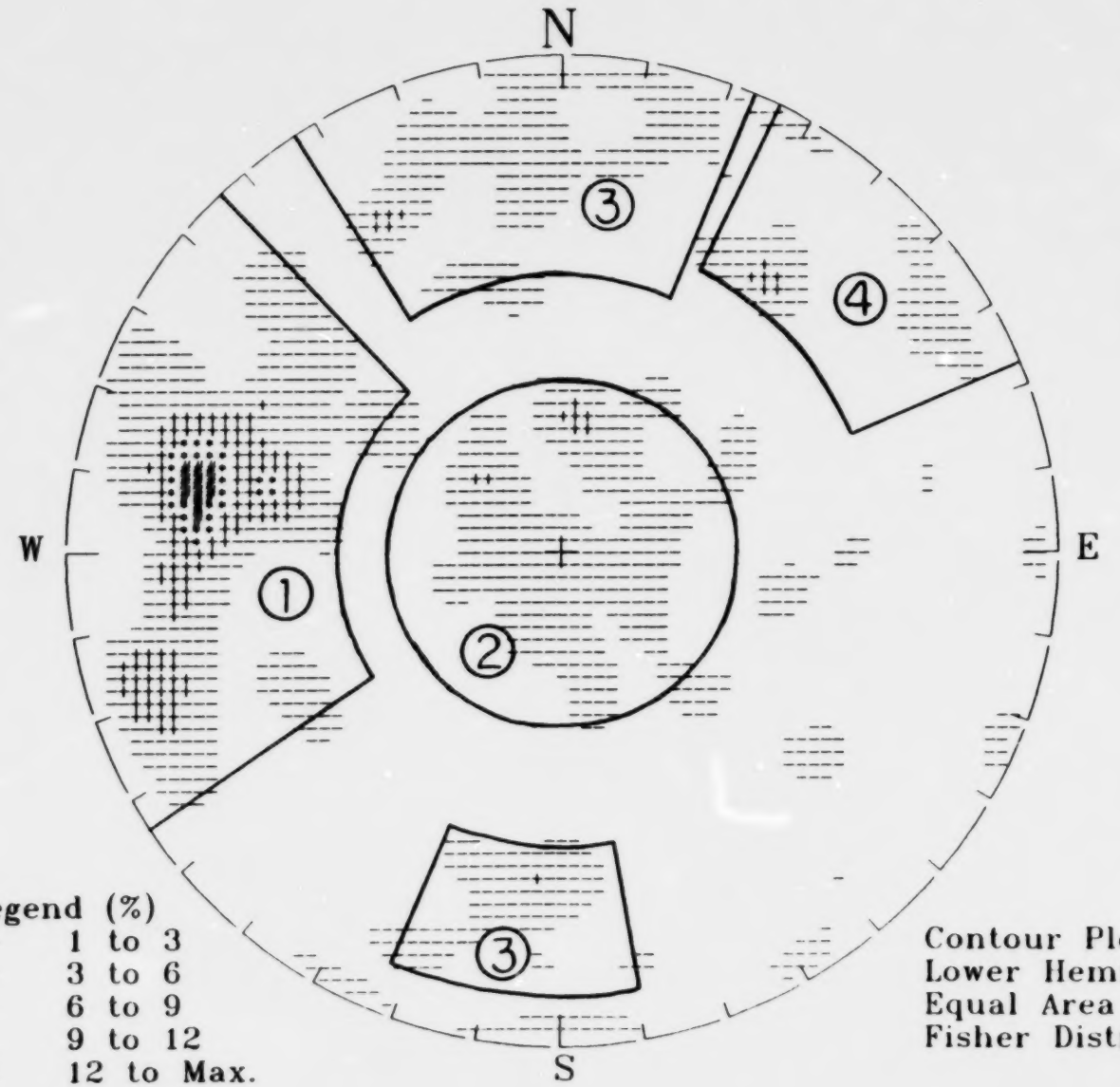
Legend (%)

----	1 to 5
++++	5 to 10
••••	10 to 15
####	15 to 20
====	20 to Max.

Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

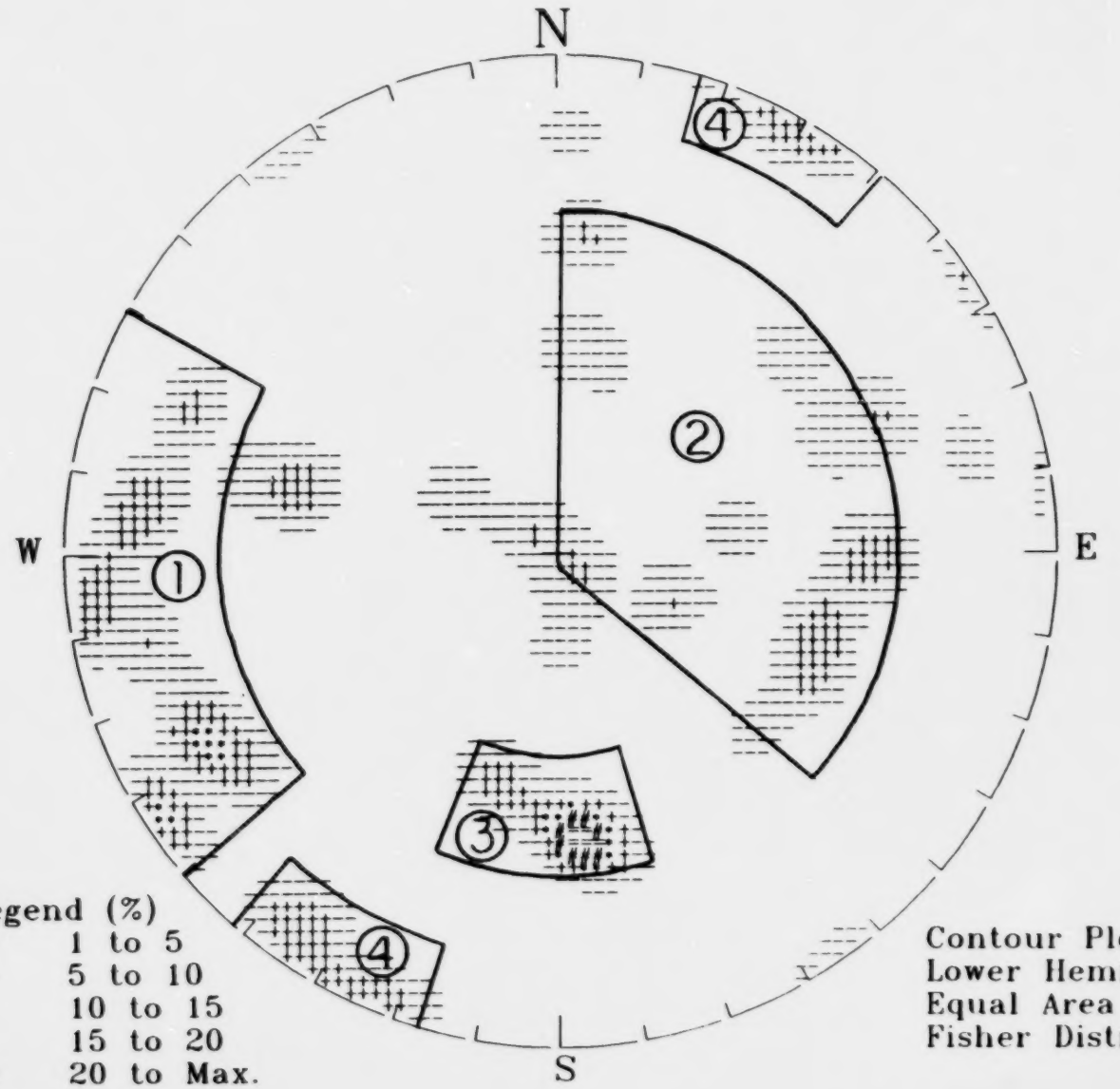
STEREONET – 1A ZONE

FIG. 2.10



STEREONET – 1B ZONE

FIG. 2.11



Legend (%)

----	1 to 5
++++	5 to 10
****	10 to 15
####	15 to 20
====	20 to Max.

Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

STEREONET – 1C ZONE

FIG. 2.12

Legend (%)

----	1 to 3
++++	3 to 6
****	6 to 9
####	9 to 12
====	12 to Max.

Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

STEREONET – THOMPSON MINE

FIG. 2.13

JOINT SET ORIENTATION BY ORE ZONE THOMPSON MINE

FIG. 2.14

(Dip/Dip Direction)

ZONE	JOINT SET 1	JOINT SET 2	JOINT SET 3	JOINT SET 4	JOINT SET 5	JOINT SET 6
Nose	85/225	15/350	–	–	–	–
2 Shear	82/50	11/320	70/170	–	63/115	–
3 Shear	80/95	30/295	85/170	85/45	–	–
1A	69/80	16/240	66/170	63/235	–	–
1B	69/100	10/235	80/180	71/225	–	–
1C	70/85	30/260	–	85/30	–	48/0

ROCK STRENGTH DESCRIPTION

Golder Assoc. Hardness Code	Geological Society of London Term	Approx. Uniaxial Compressive Strength MPa	psi	Field Estimation of Hardness*
R5	Extremely strong	> 200	> 29,200	Requires many blows of geological hammer to break.
R4	Very strong	100 – 200	14,600 – 29,200	Requires a few blows of geological hammer to break.
R3	Strong	50 – 100	7,300 – 14,600	Breaks under single blow of geological hammer.
R2	Moderately strong	12.5 – 50	1,825 – 7,300	1/4" indentations with sharp end of geological pick.
	Moderately weak	5.0 –12.5	730 – 1,825	Too hard to cut by hand into tri–axial specimen.
R1	Weak	1.25 – 5.0	182 – 730	Crumbles under firm blows of geological pick.
S5	Very weak (rock)	0.60 – 1.25	85 – 182	May be broken in the hand with difficulty.
S4	Very stiff (soil)	0.30 – 0.60	42 – 85	Indented by finger nail.
S3	Stiff	0.15 – 0.30	21 – 42	Cannot be moulded in fingers.
S2	Firm	0.08 – 0.15	10.5 – 21	Moulded with strong pressure of fingers.
S1	Soft	0.04 – 0.08	5.25 – 10.5	Easily moulded with fingers.

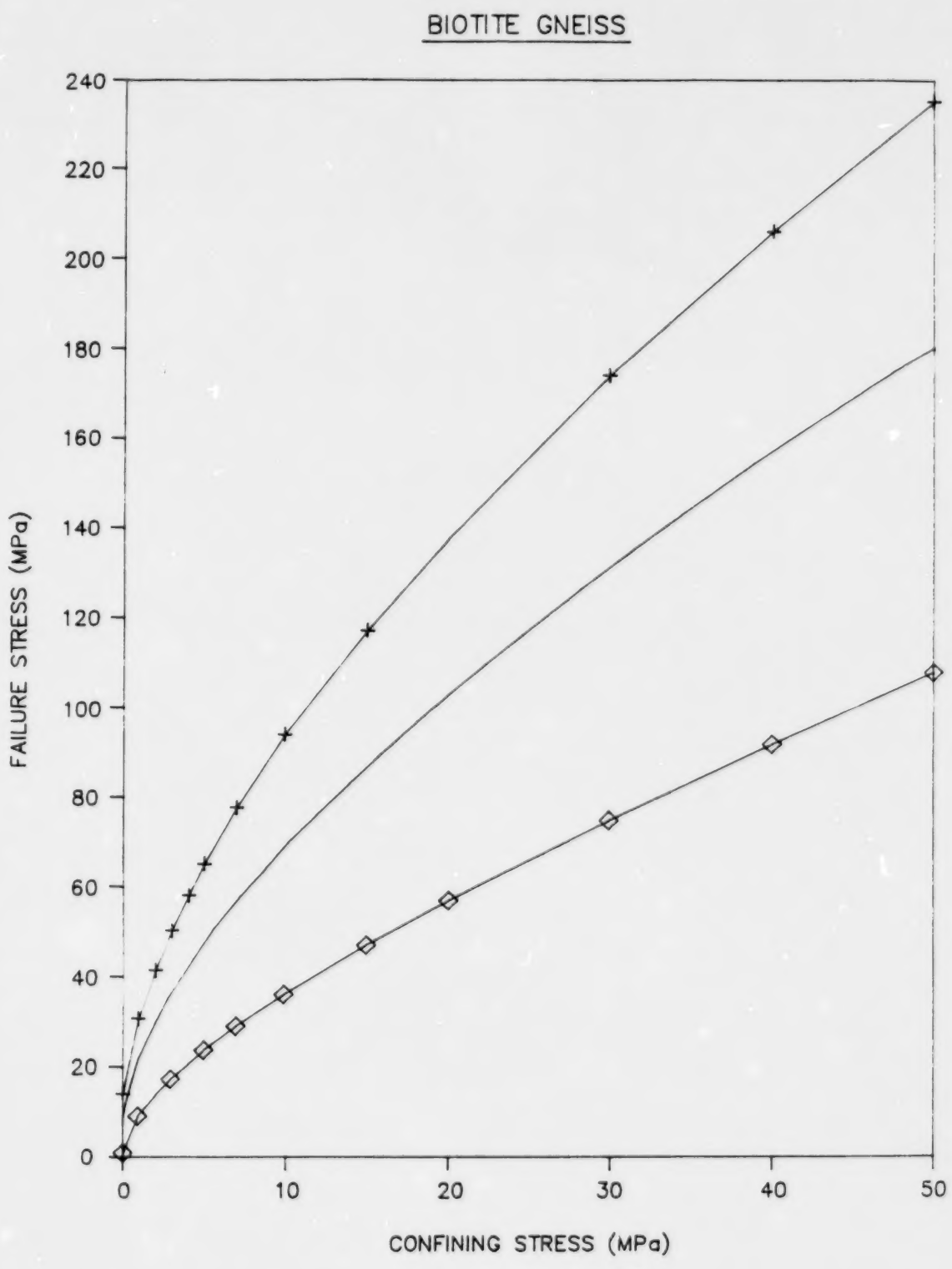
*All field estimates based on a hand held specimen of rock or rock core.

ROCK STRENGTH DESCRIPTION

FIG. 2.15

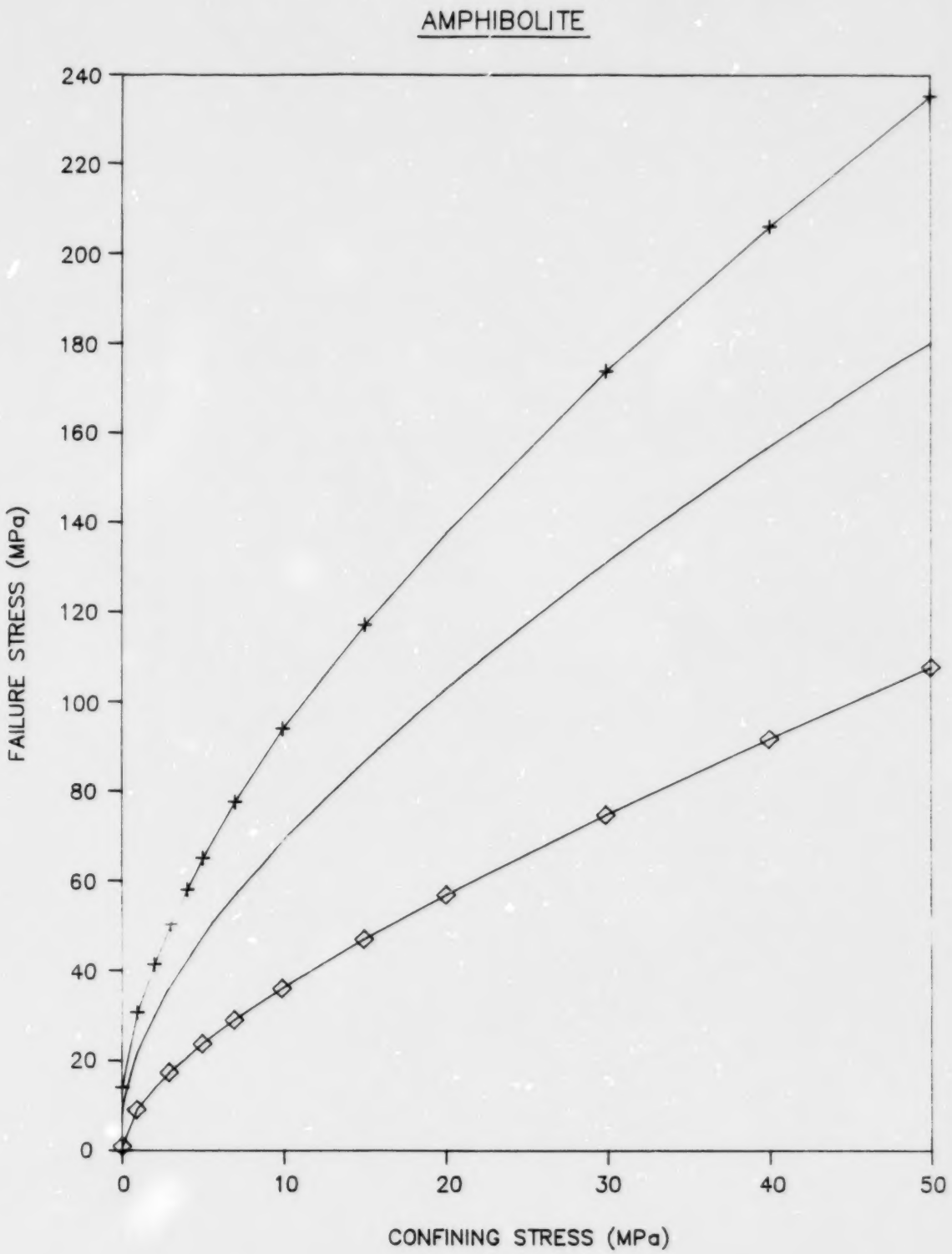
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.16



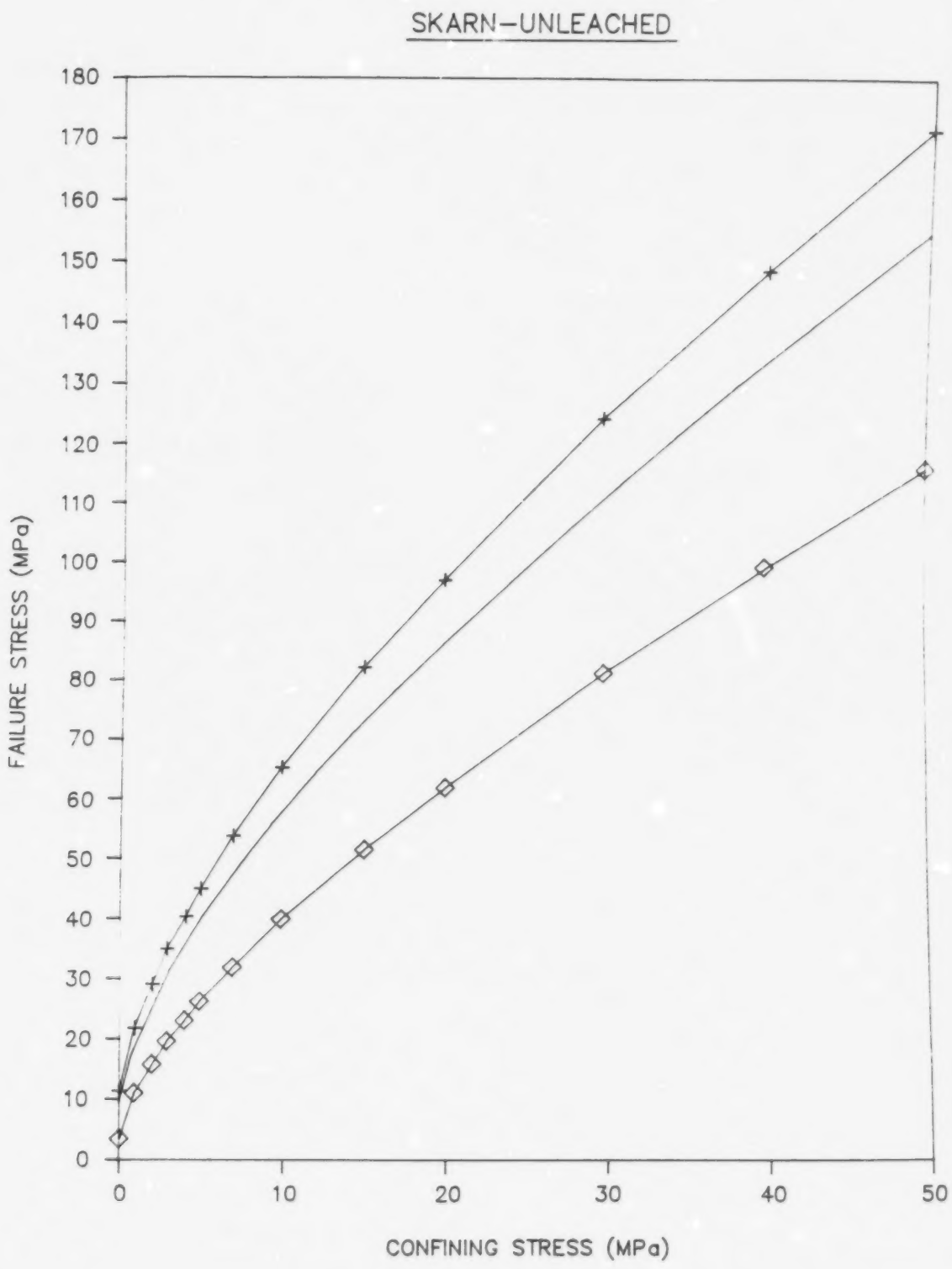
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.17



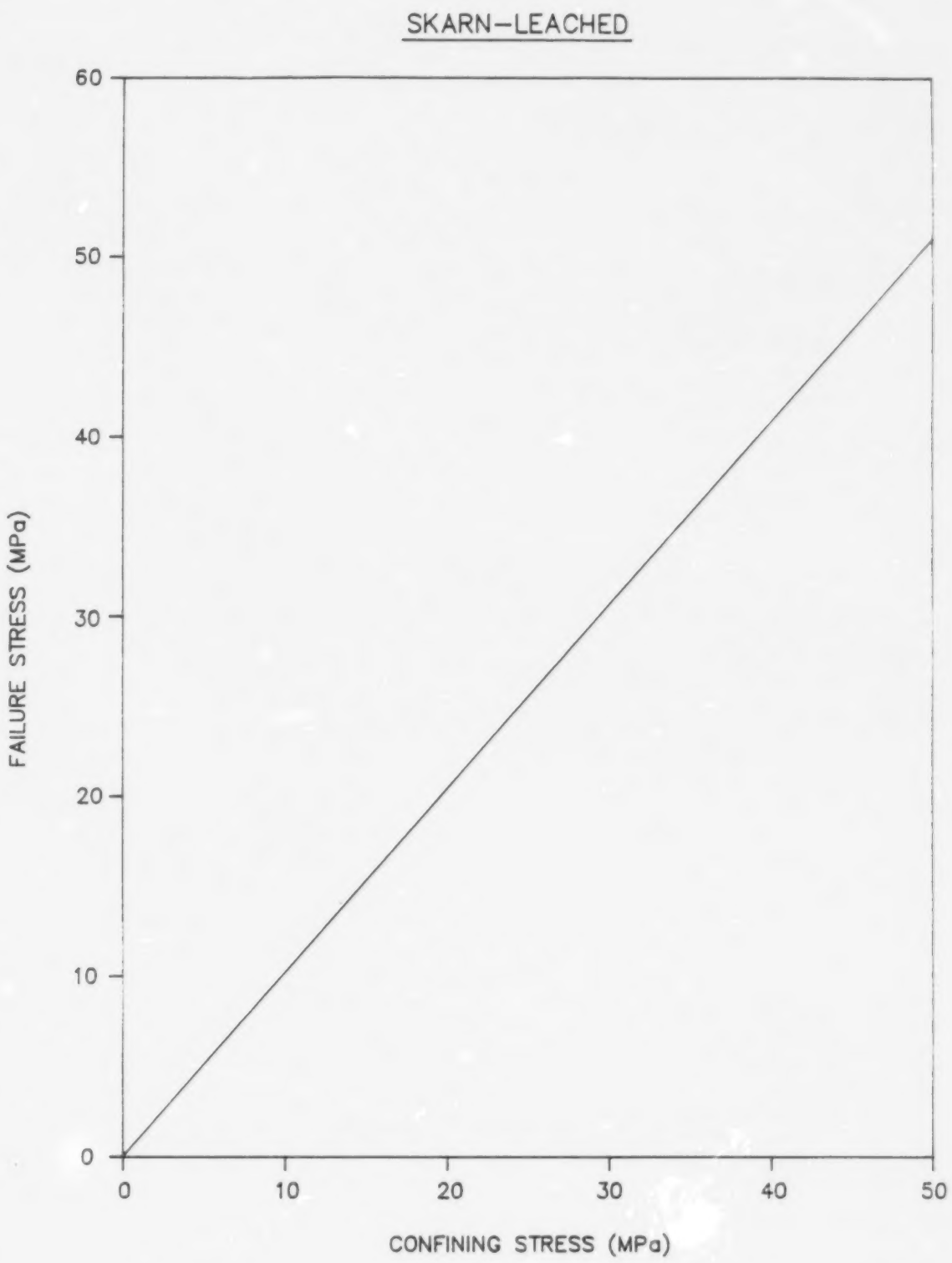
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.18



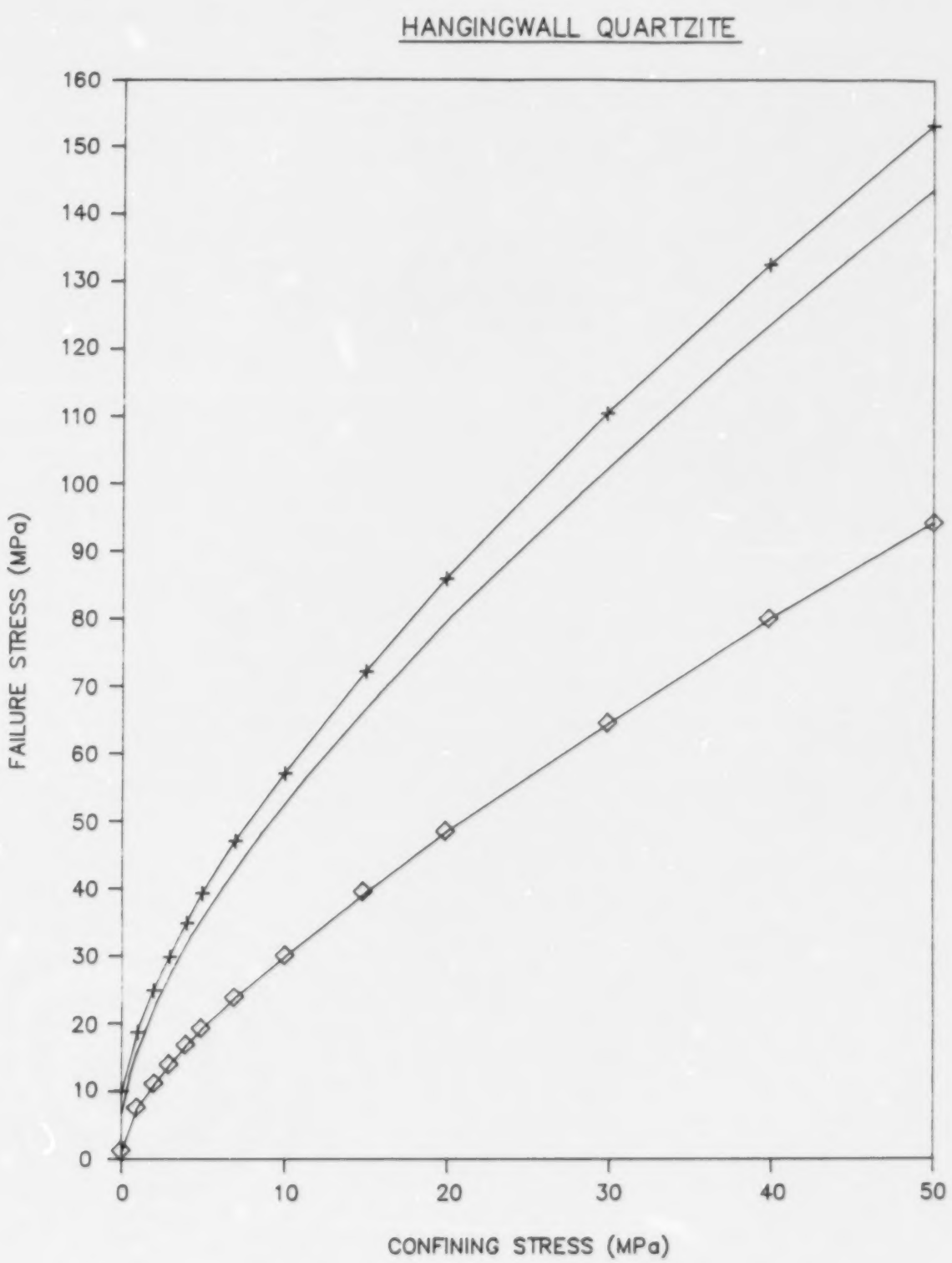
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.19



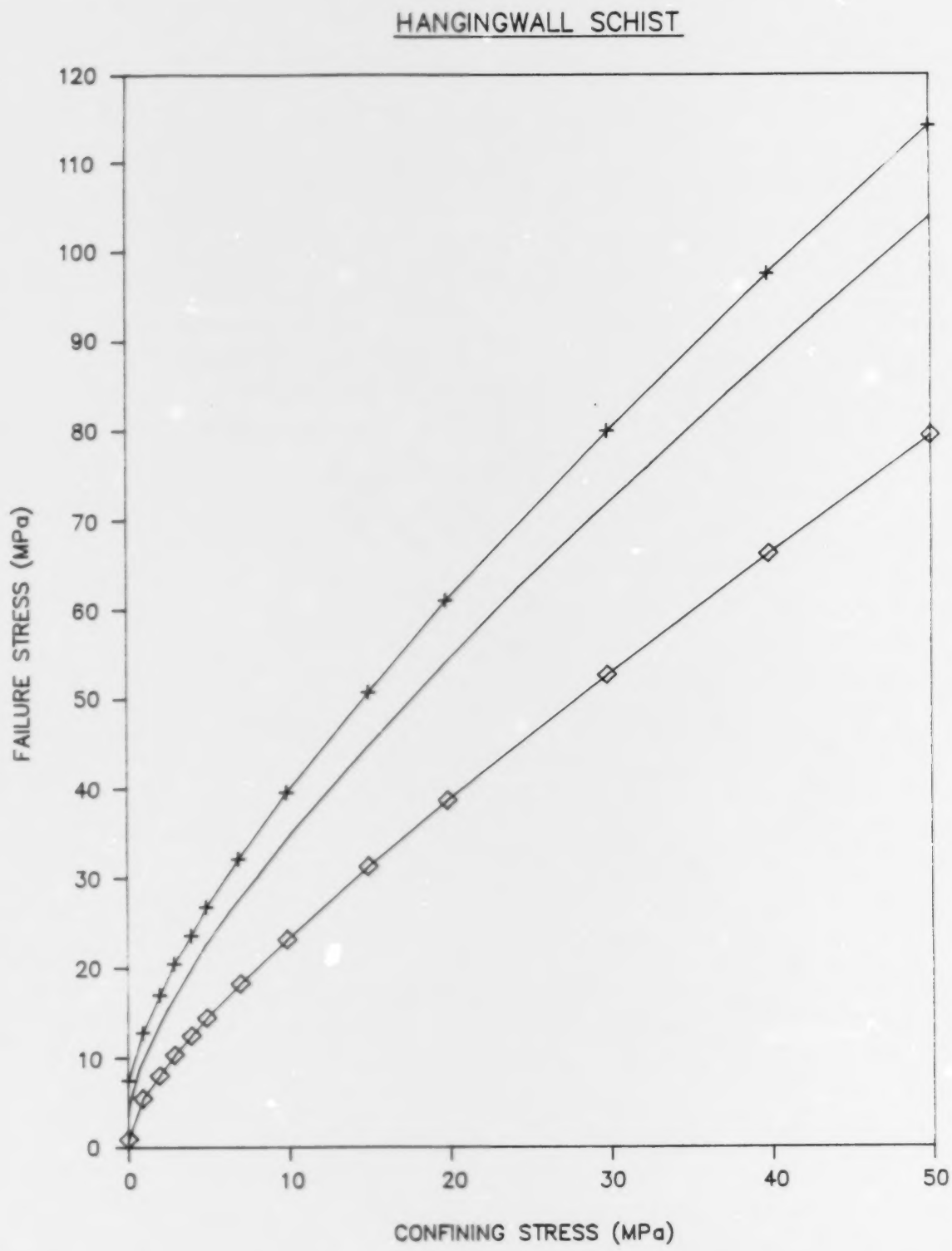
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.20



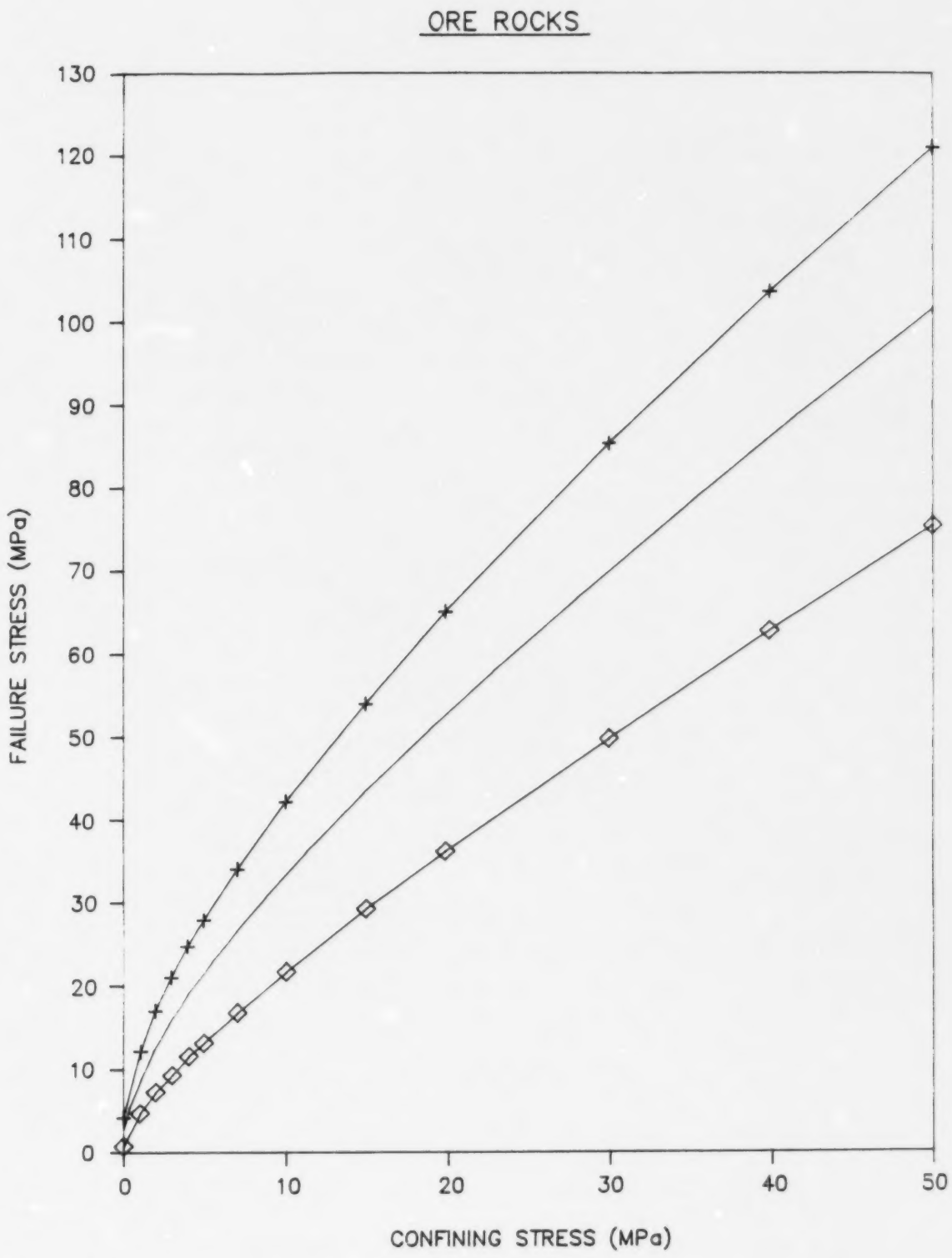
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.21



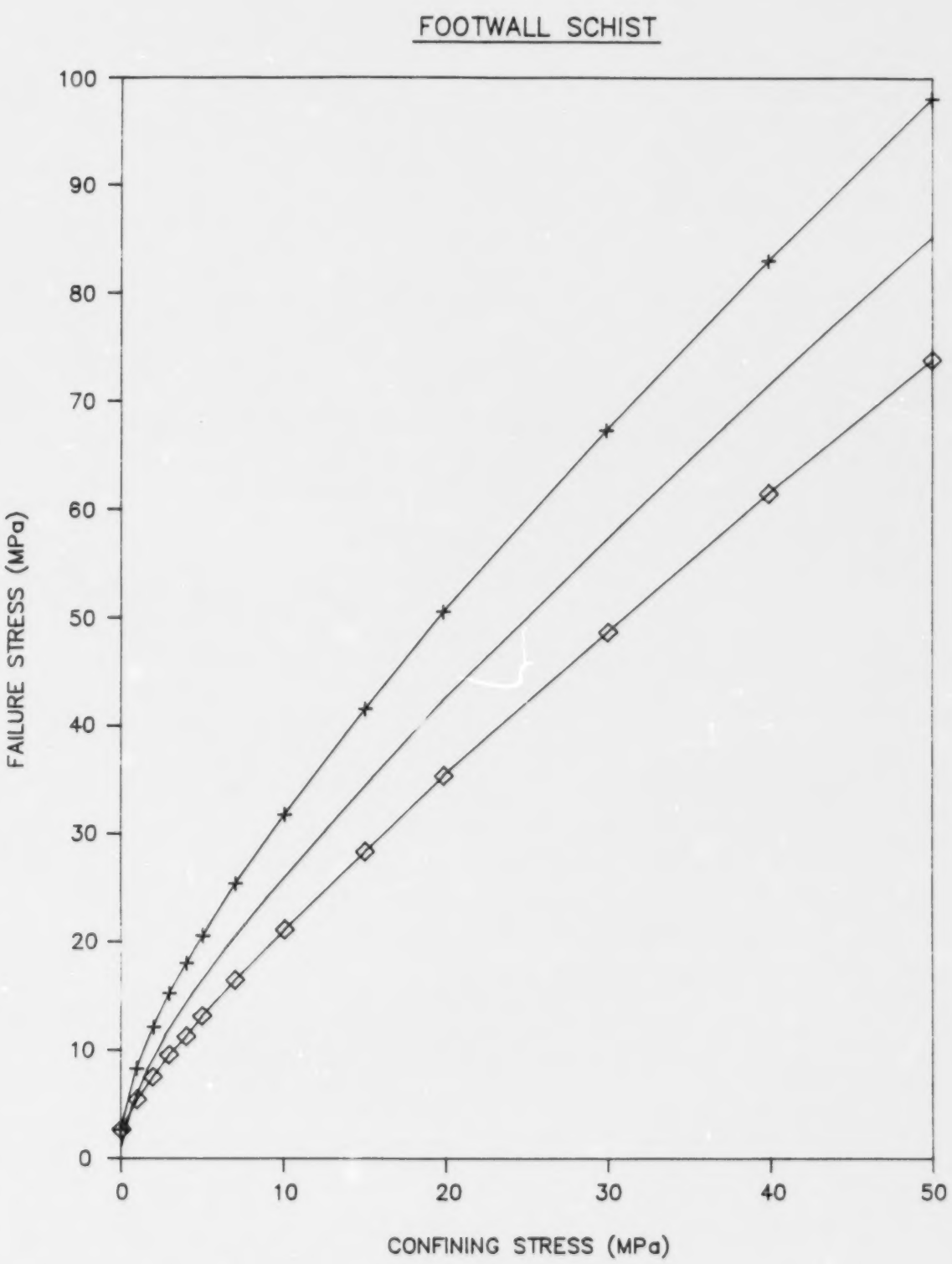
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.22



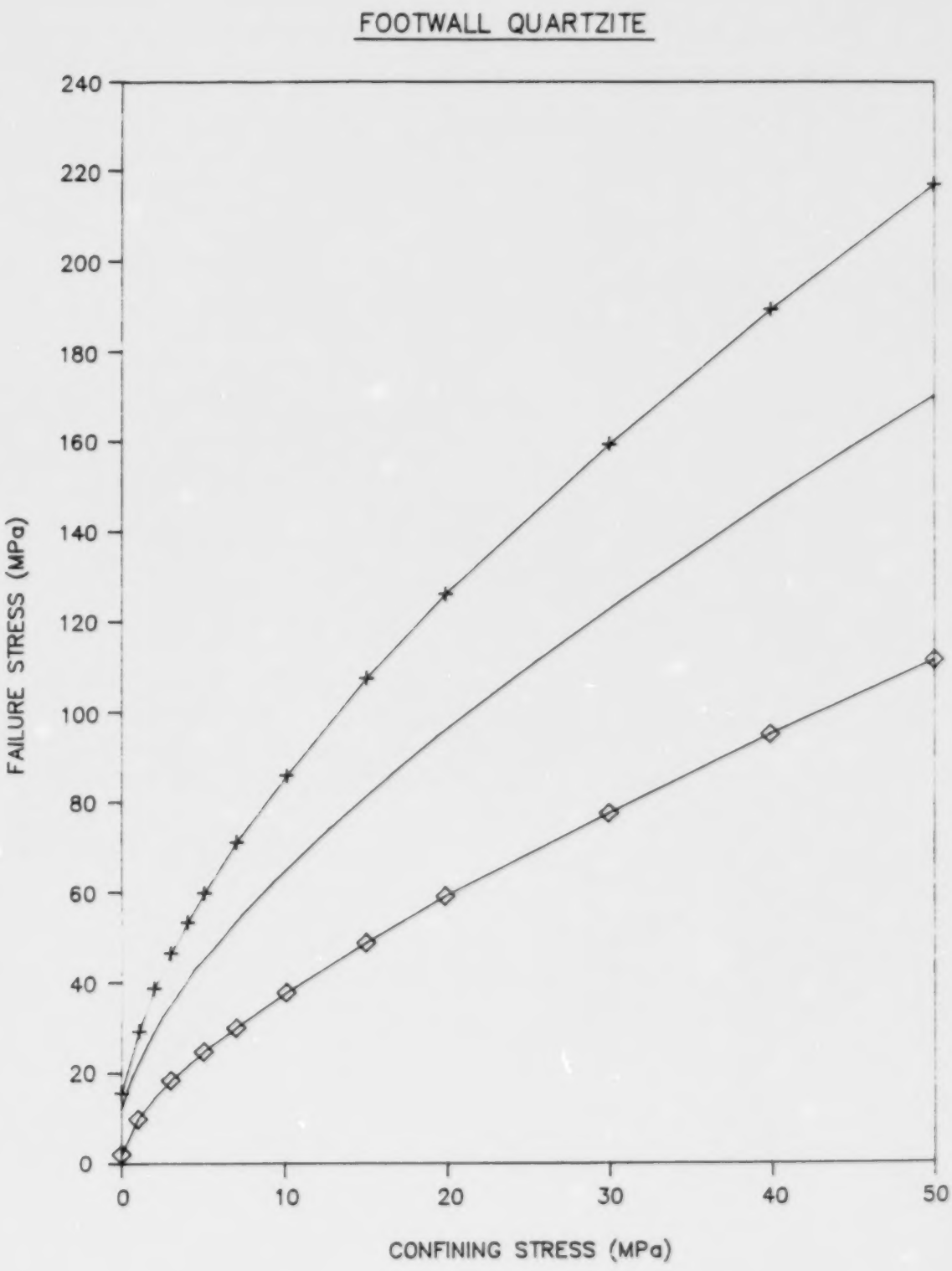
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.23



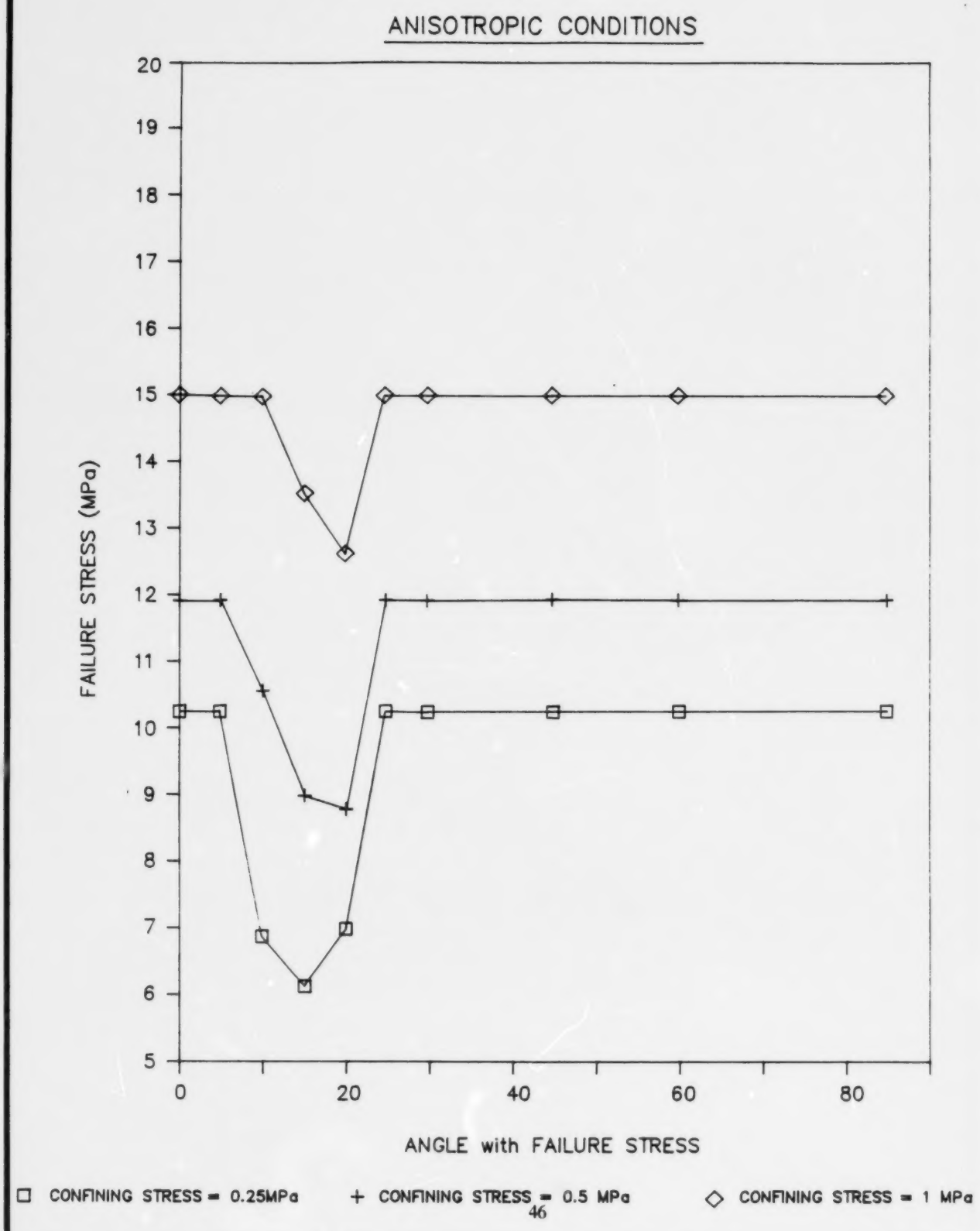
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.24



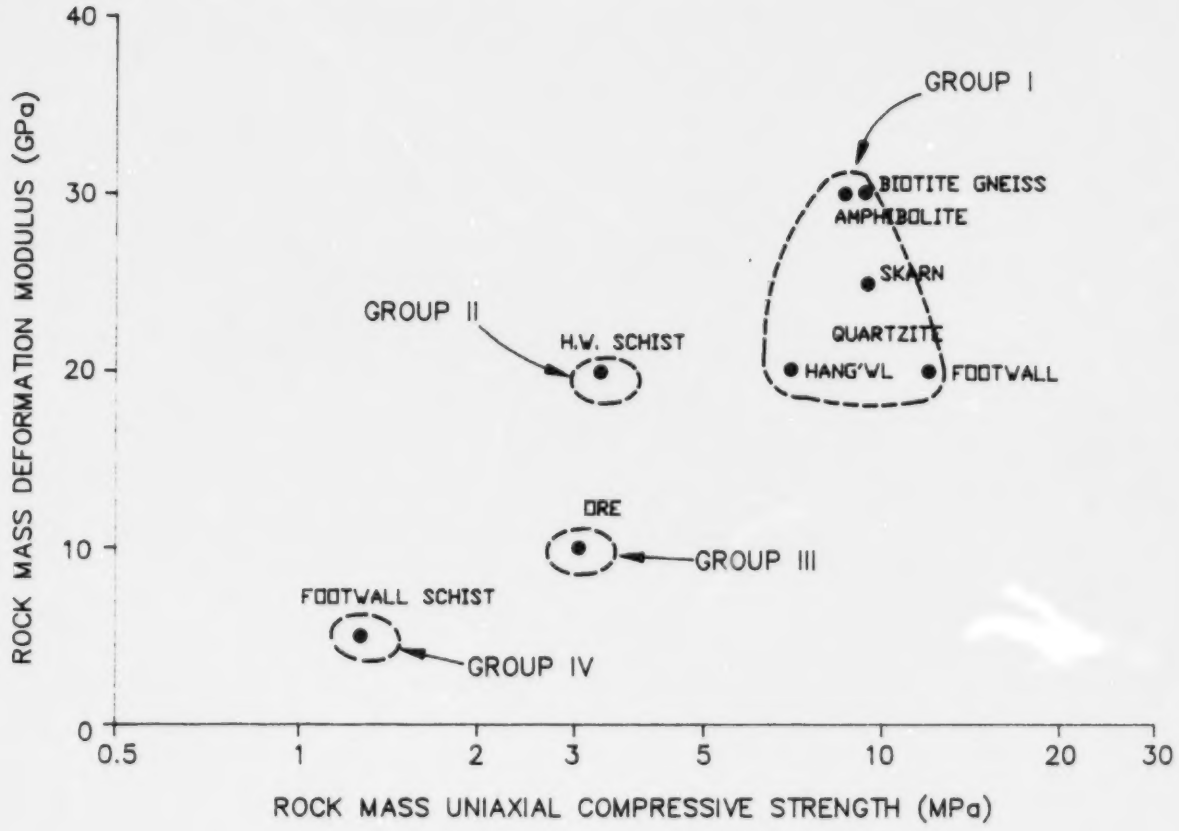
ESTIMATION OF ROCK MASS STRENGTH

FIG. 2.25



CLASSIFICATION ROCK MASS PROPERTIES

FIG. 2.26



CLASSIFICATION ROCK MASS PROPERTIES

UNIAXIAL COMPRESSIVE STRENGTH TESTS

FIG. 2.27

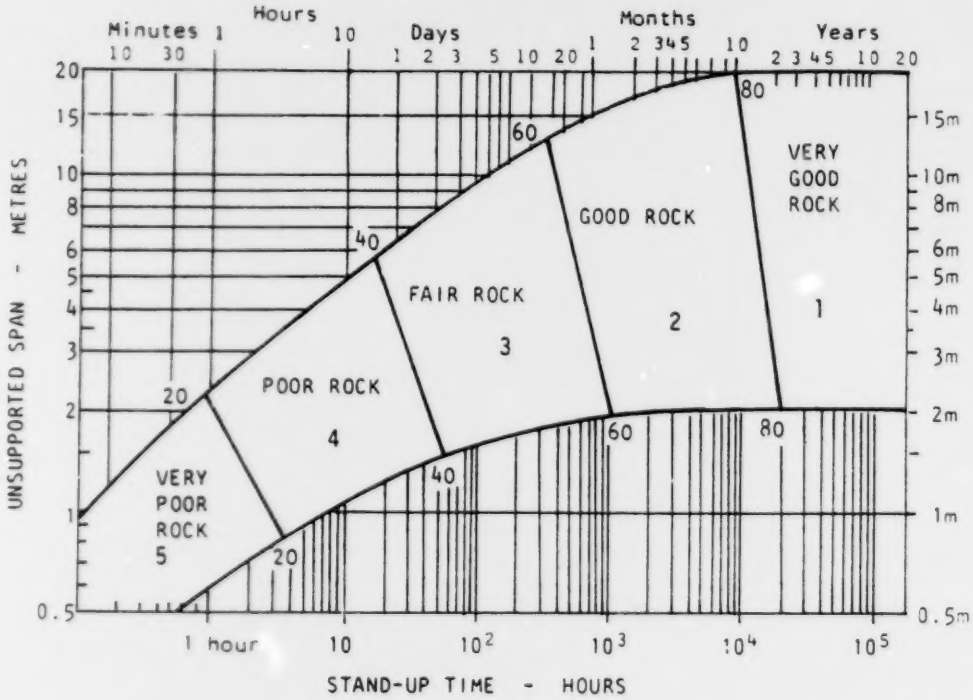
Summary of Uniaxial Compressive Strength Tests

STRUCTURE	ROCK TYPE	UNIAXIAL COMPRESSIVE STRENGTH (MPa)
Hangingwall	Quartzite	135
	Biotite Gneiss	150
	Amphibolite	139
	Ultramafic Amphibolite	250
	Skarn (Unleached)	153
	Biotite Schist	1. 70 2. 138
Footwall	Quartzite	204
	Biotite Gneiss	1. 70 2. 138
Orebody	Mineralized Schist	66
	Massive and Brecciated Sulphide	72

NOTES: 1. P<55° to Foliation
2. P>70° to Foliation

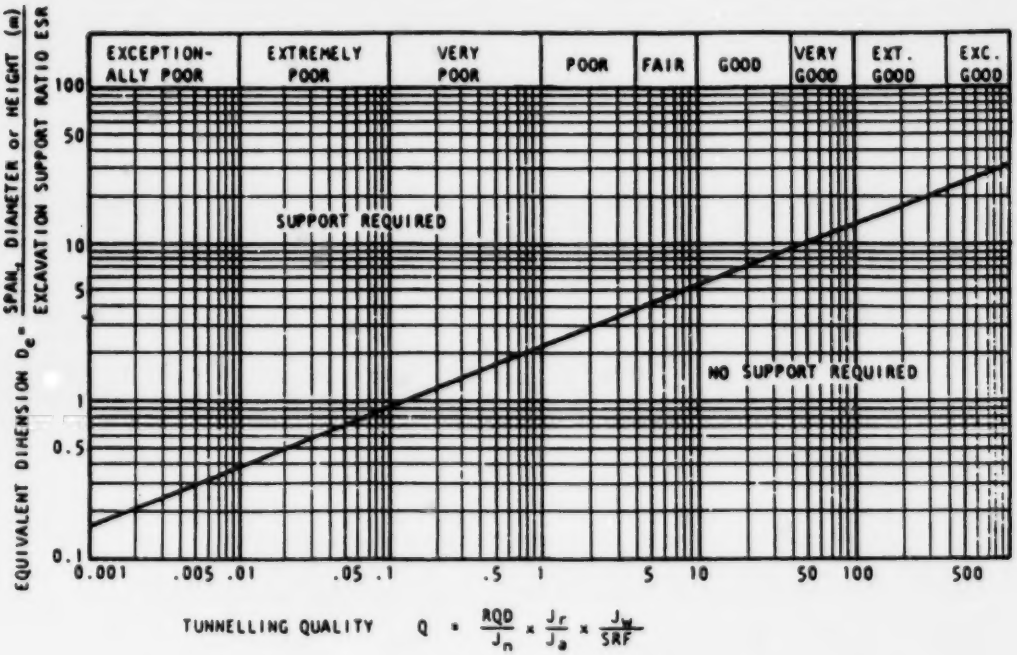
RELATION BETWEEN STAND-UP TIME, EXCAVATION SPAN AND CSIR CLASSIFICATION

FIG. 2.28



ROCK MASS RATING SUPPORT CLASSIFICATION

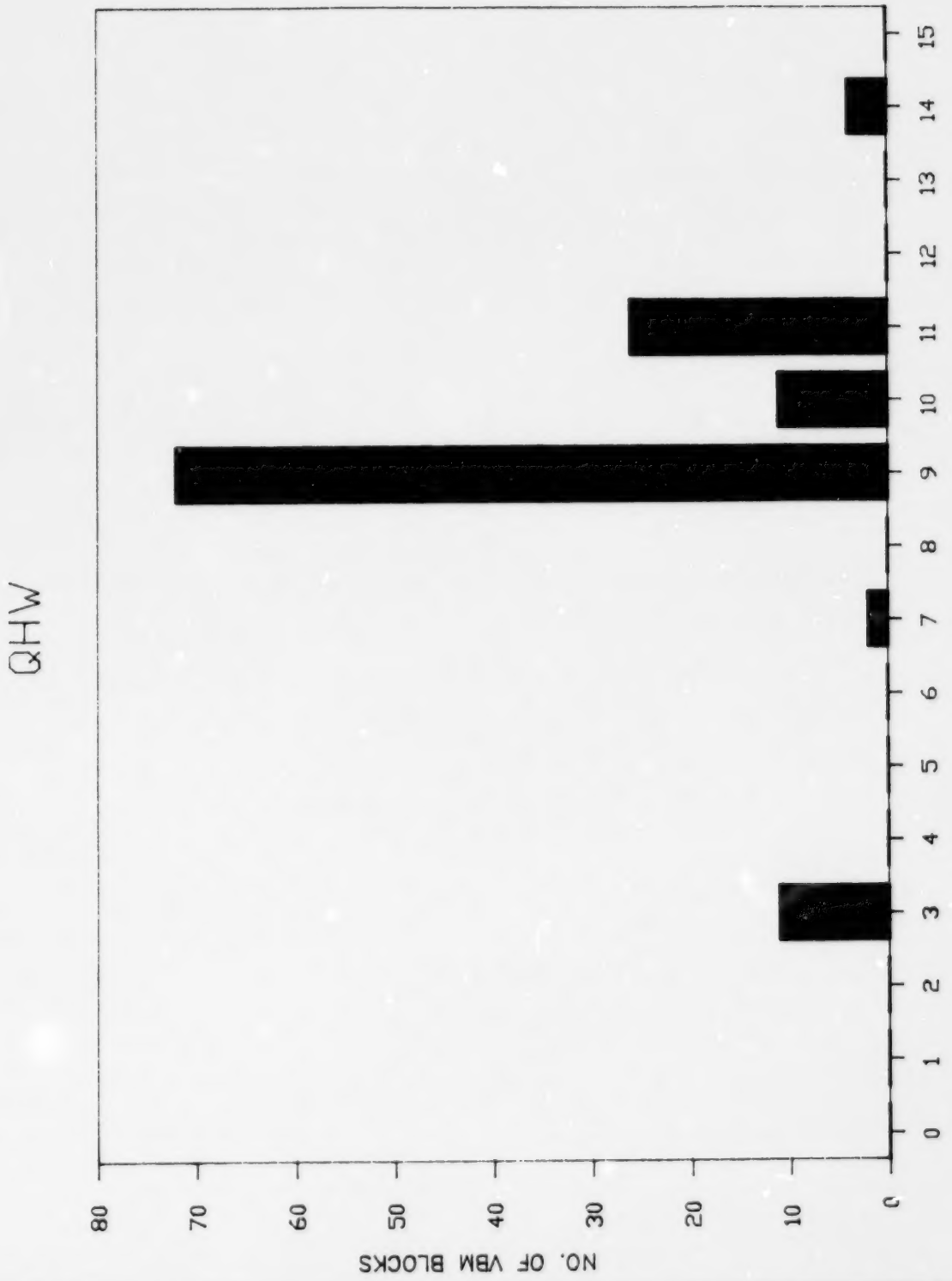
FIG. 2.29



Relationship between the maximum equivalent dimension D_e of an unsupported underground excavation and the NGI tunnelling quality index Q.
(After Barton, Lien and Lunde[1])

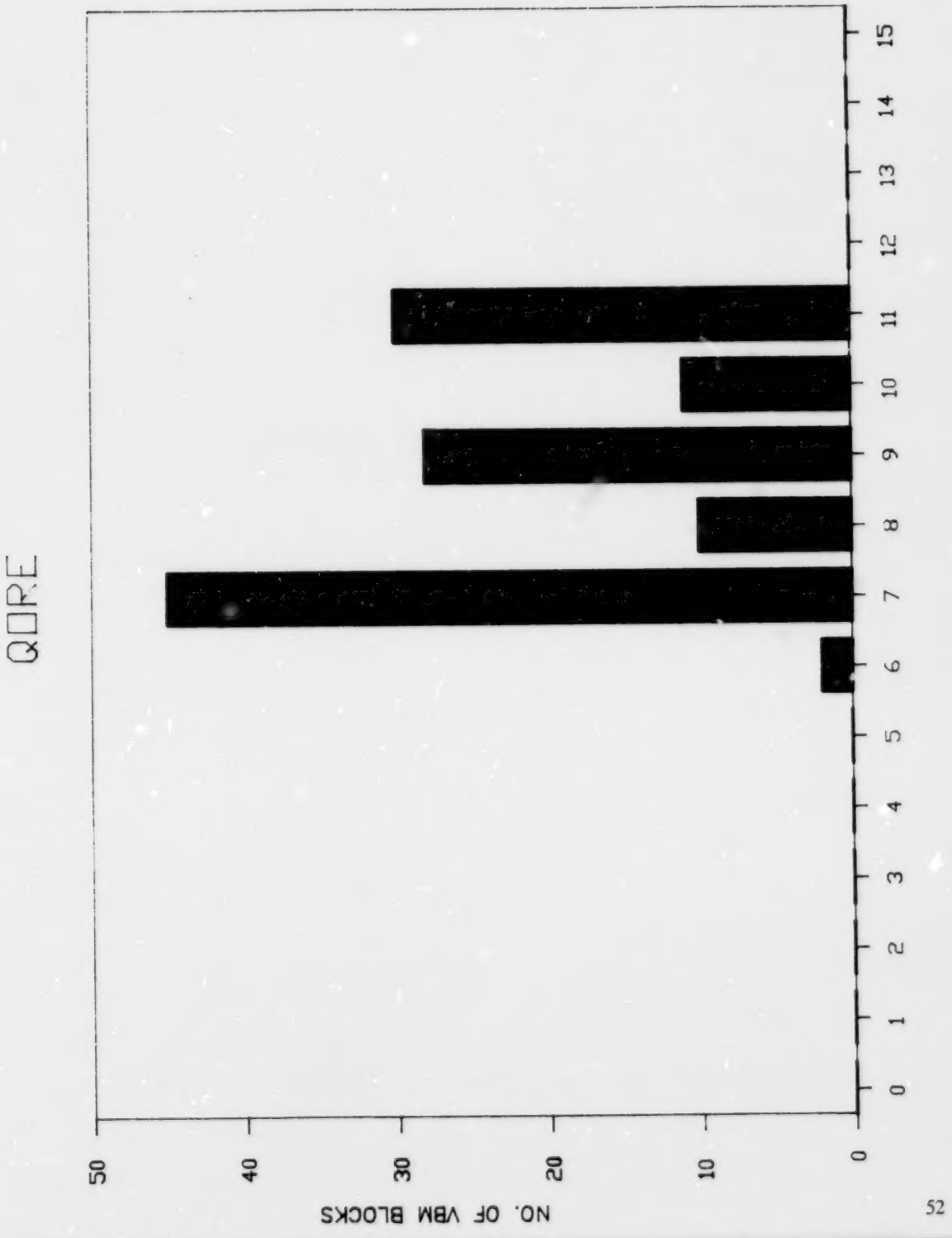
HANGINGWALL QUALITY

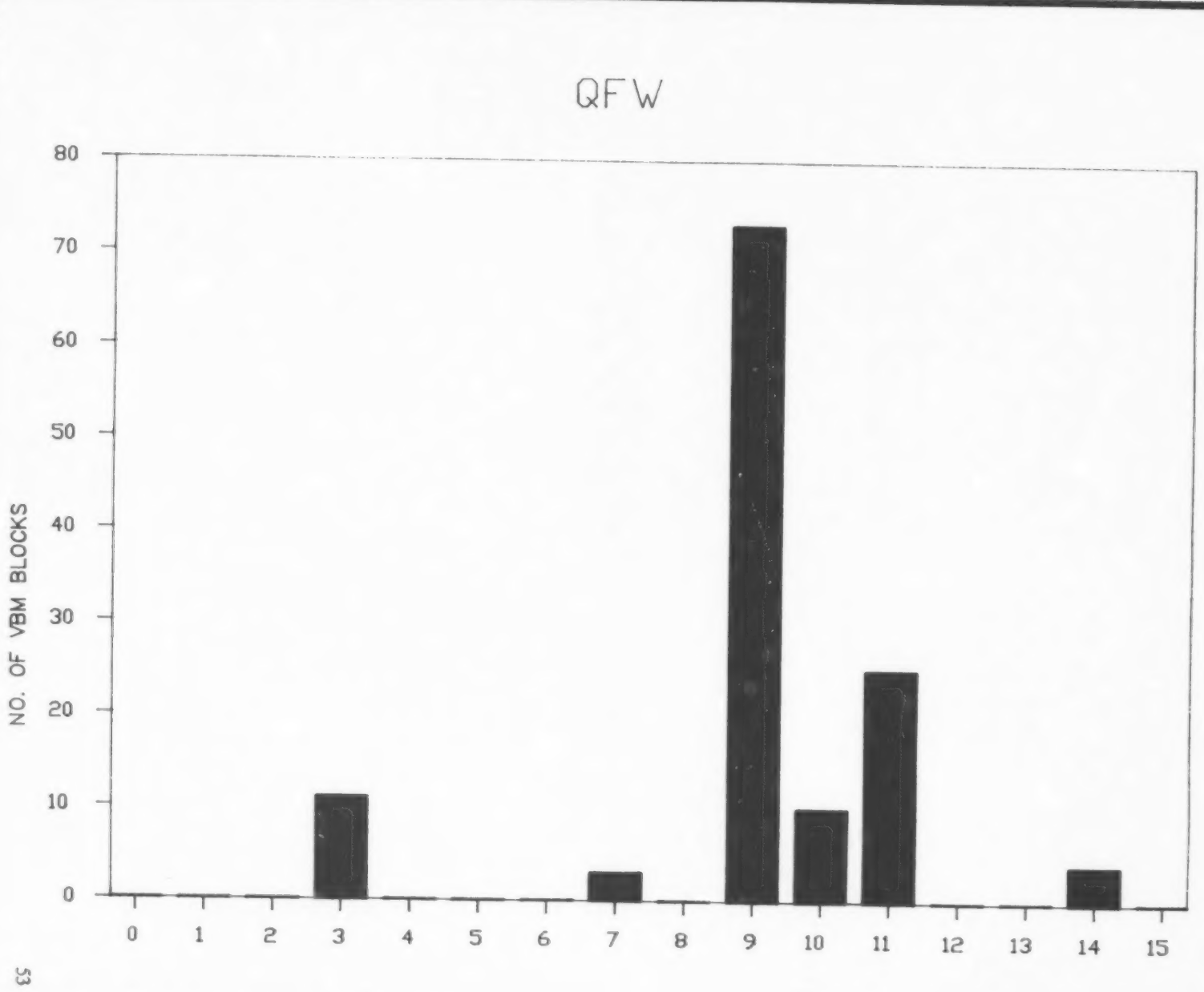
FIG. 2.30



ORE QUALITY

FIG. 2.31



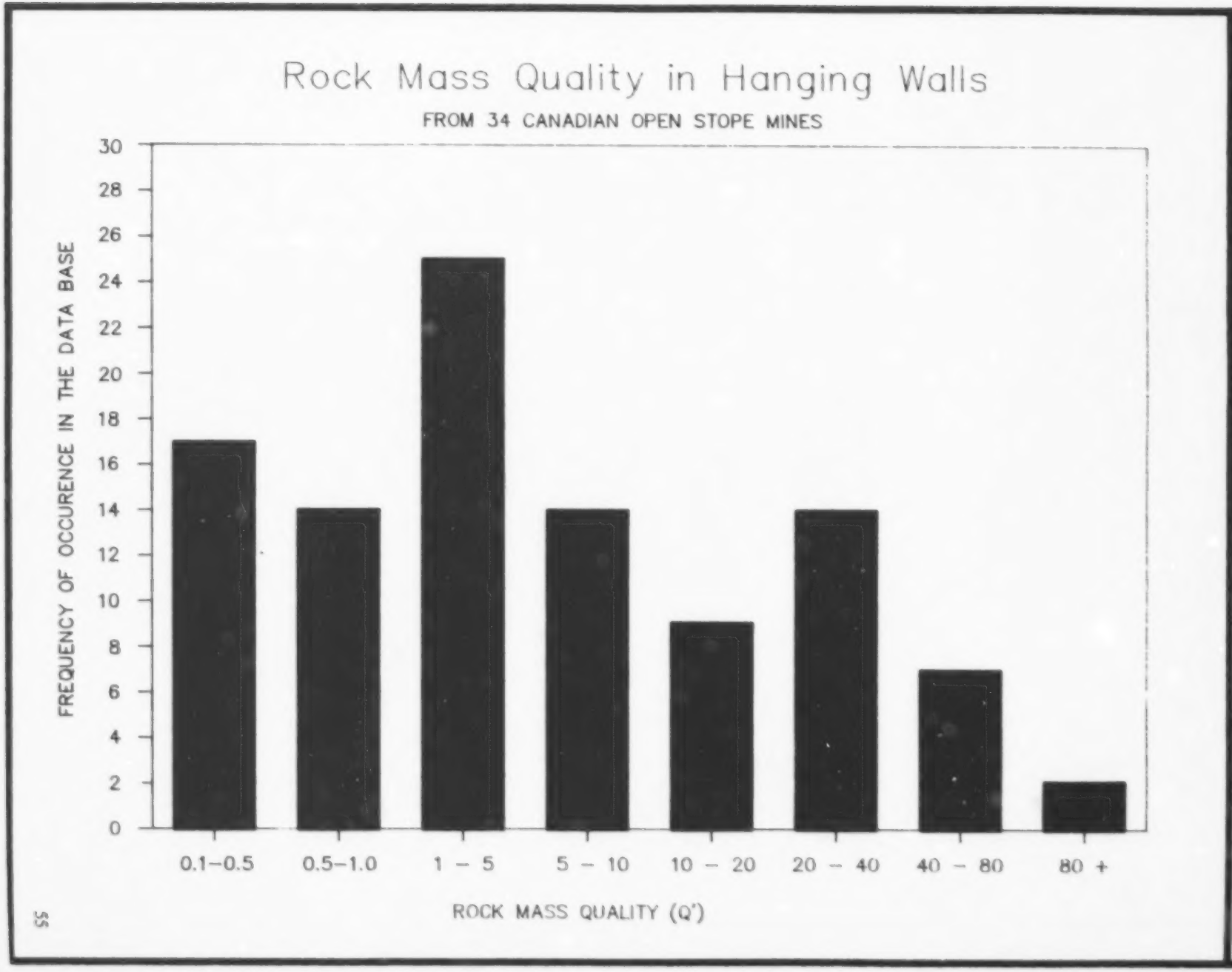


FOOTWALL QUALITY

FIG. 2.32

ROCK MASS QUALITY IN OPEN STOPE BACKS

FIG. 2.33

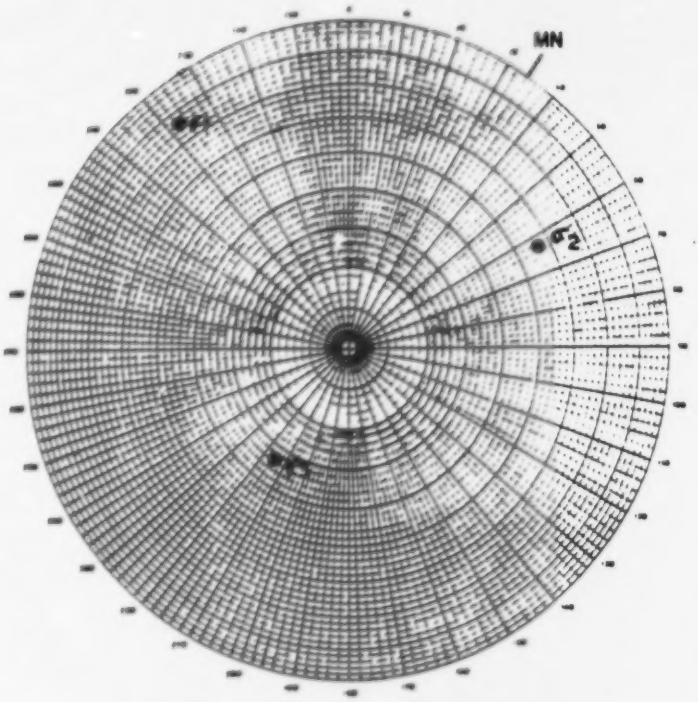


ROCK MASS QUALITY IN HANGING WALLS

FIG. 2.34

RESULTS OF CANMET STRESS MEASUREMENTS

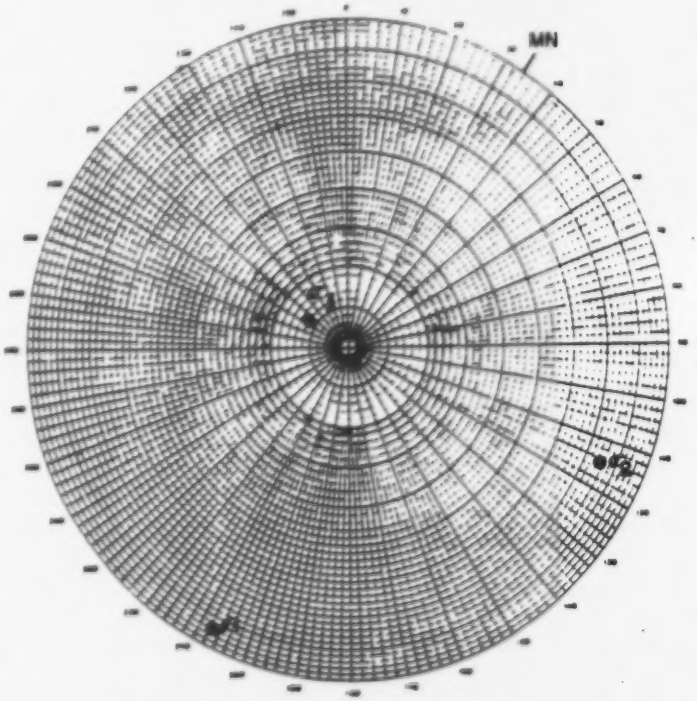
FIG. 2.35



POLAR EQUAL-AREA STEREO-NET

σ_1 = 58.5 MPa
σ_2 = 40.5 MPa
σ_3 = 18.3 MPa

RESULTS OF STRESS MEASUREMENT -2000 LEVEL



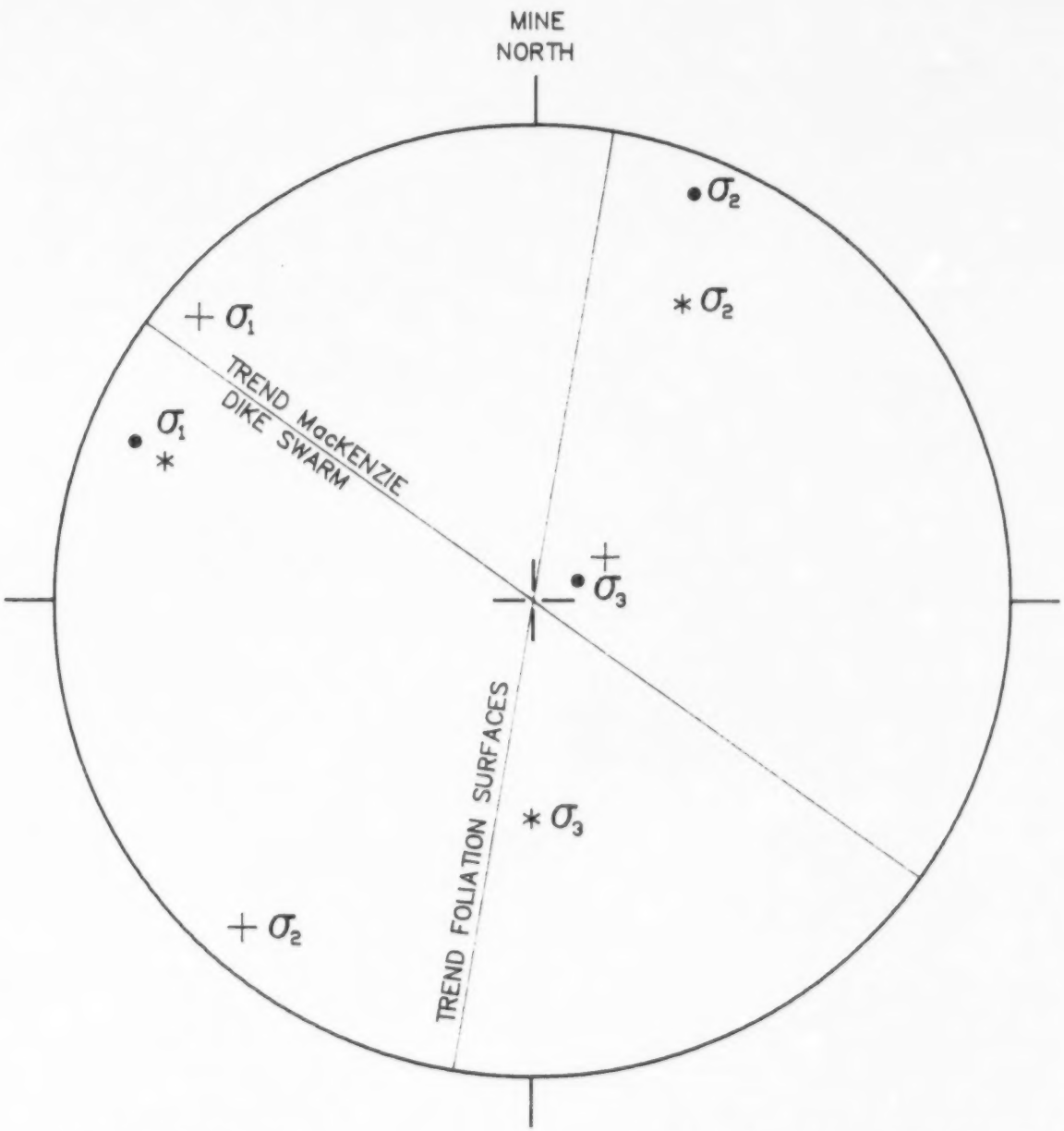
POLAR EQUAL-AREA STEREO-NET

σ_1 = 113.4 MPa
σ_2 = 112.7 MPa
σ_3 = 96.6 MPa

RESULTS OF STRESS MEASUREMENT -4000 LEVEL

STRESS ORIENTATION VS. GEOLOGICAL STRUCTURES

FIG. 2.36



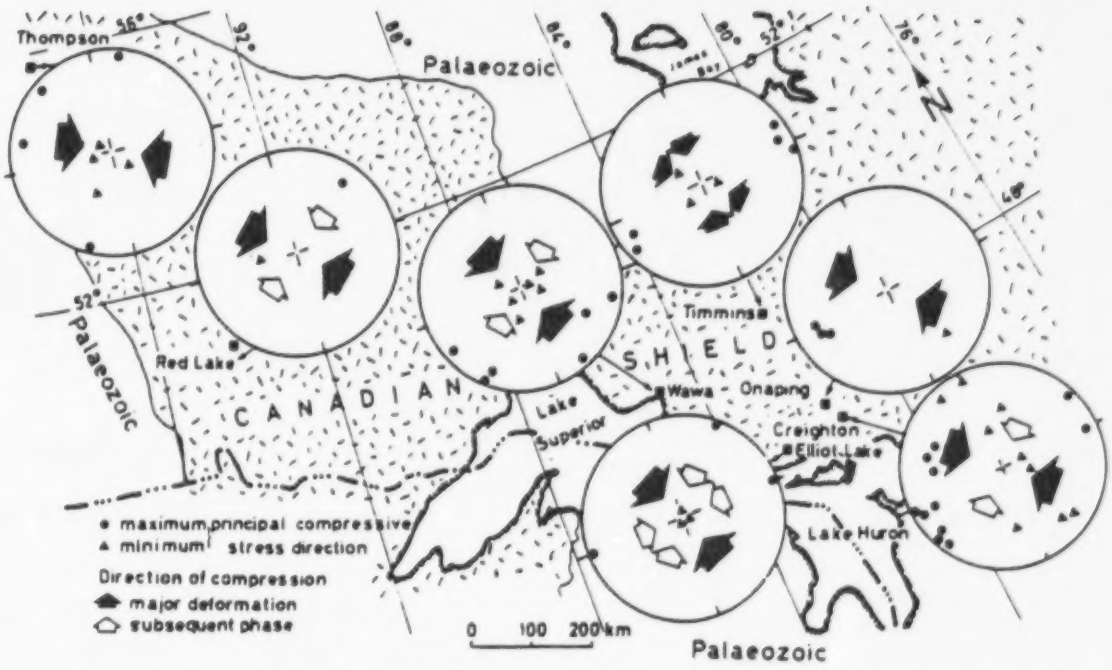
LOWER HEMISPHERE
POLAR EQUAL AREA
STEREO–NET

- AV. 400 LEVEL MEASUREMENT
+ AV. 1000 LEVEL MEASUREMENT
* 2000 LEVEL MEASUREMENT

VIRGIN STRESS ORIENTATION VS. TREND
MAJOR GEOLOGICAL STRUCTURES

DIRECTION OF COMPRESSION DURING TECTONIC DEFORMATION

FIG. 2.37

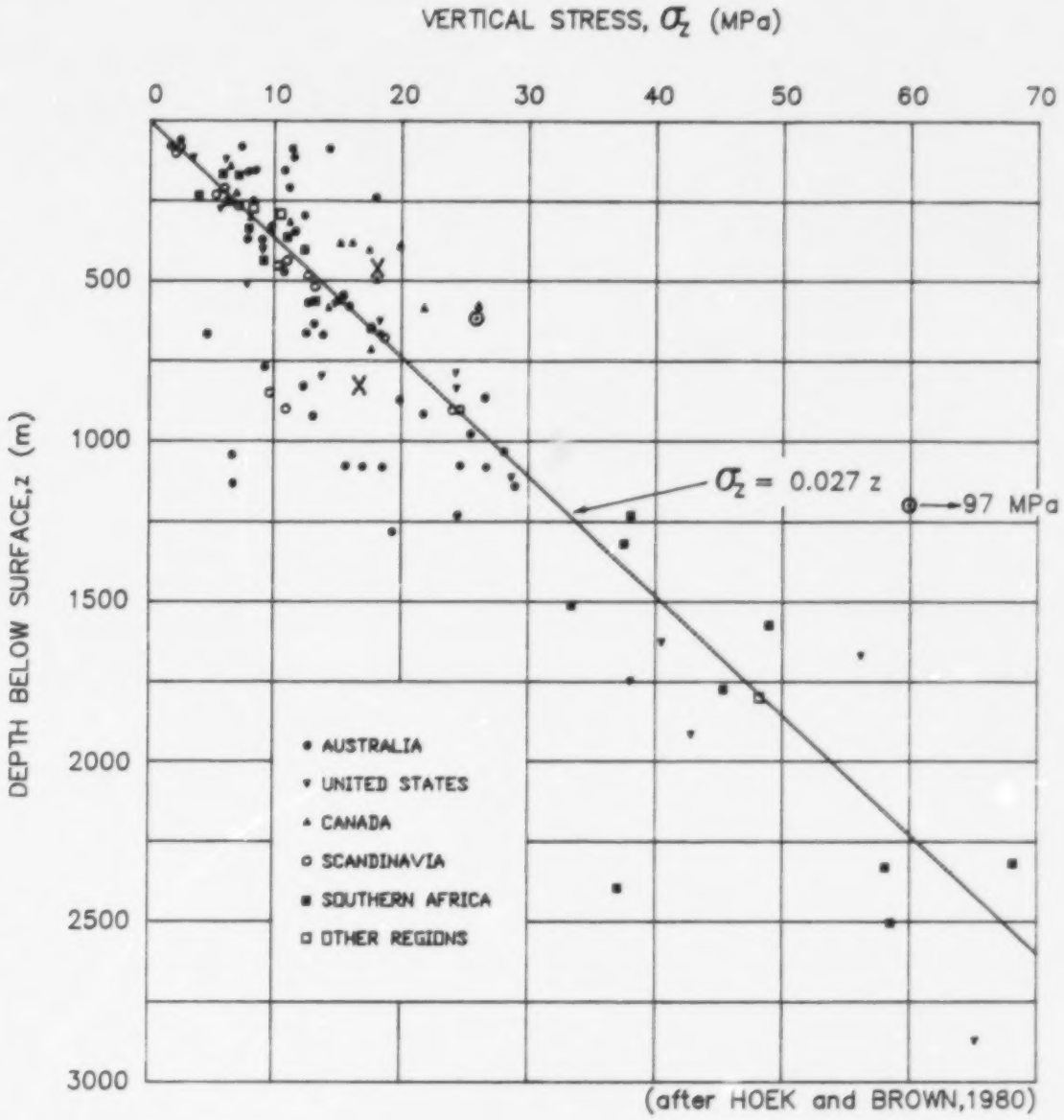


(after Hedley, 1982)

SUGGESTED DIRECTION OF COMPRESSION DURING TECTONIC DEFORMATION AND MEASURED PRINCIPAL STRESSES FROM OVERCORING PLOTTED IN LOWER HEMISPHERE OF EQUAL AREA NET

MEASURED VERTICAL STRESSES COMPARED WITH WORLD DATA

FIG. 2.38

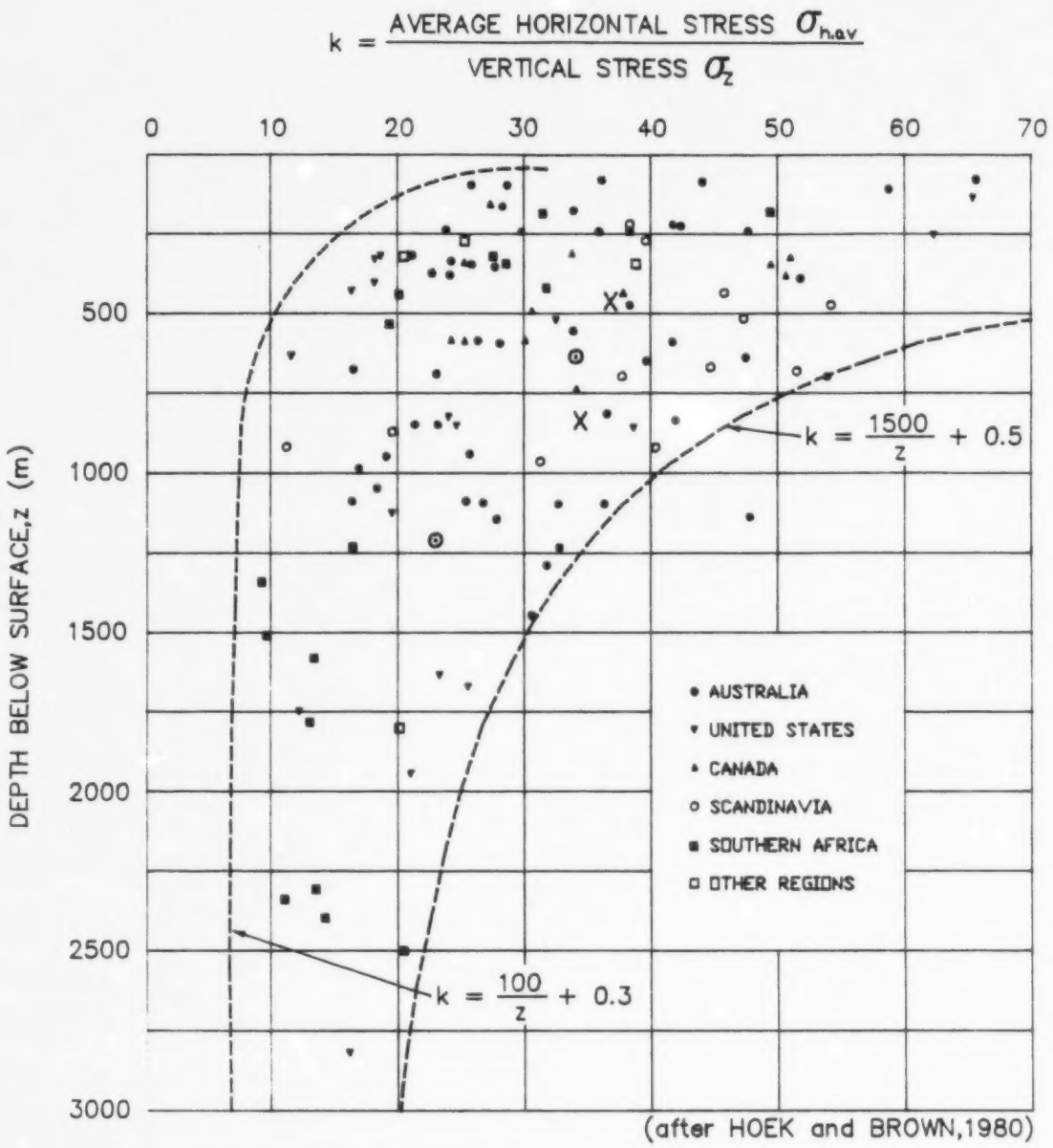


⊙ Results obtained by CANMET Thompson Mine
X Results obtained by CANMET Birchtree Mine

COMPARISON OF MEASURED VERTICAL STRESSES WITH WORLD DATA

HORIZONTAL/VERTICAL STRESS COMPARED WITH WORLD DATA

FIG. 2.39

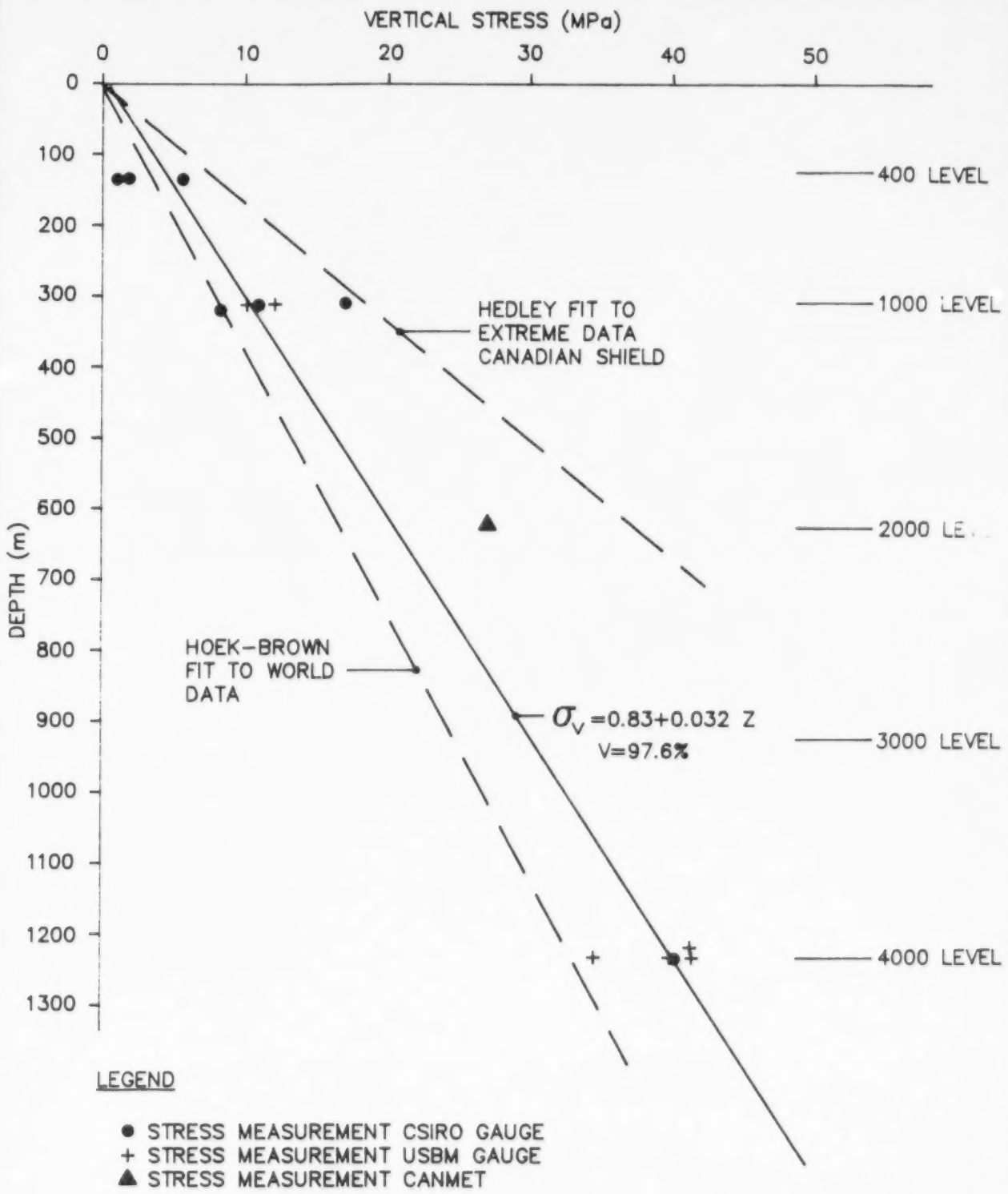


⊙ Results obtained by CANMET Thompson Mine
X Results obtained by CANMET Birchtree Mine

COMPARISON OF AVERAGE HORIZONTAL/VERTICAL STRESS RATIO WITH WORLD DATA

ESTIMATED VERTICAL STRESS GRADIENT

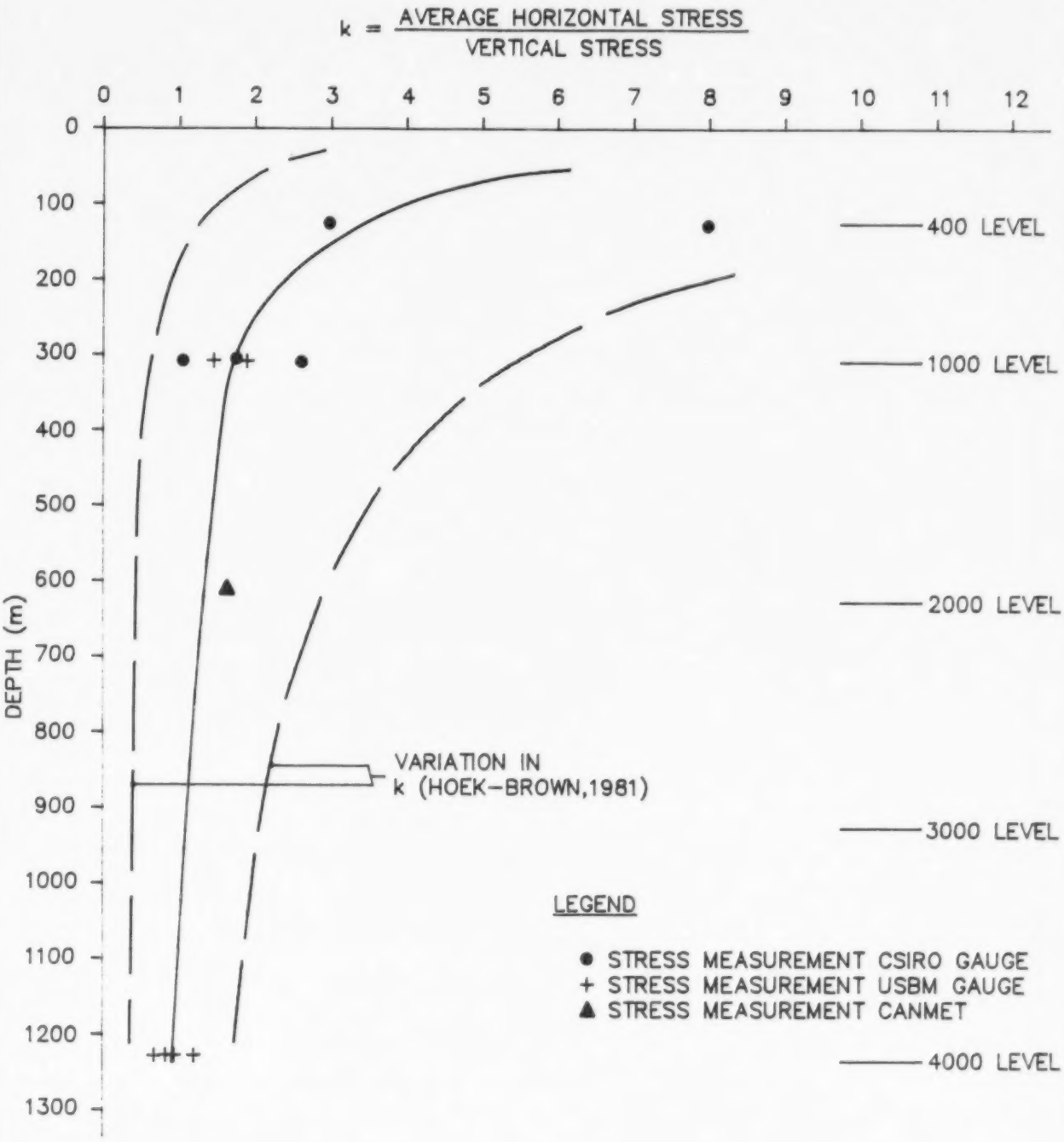
FIG. 2.40



ESTIMATED VERTICAL STRESS GRADIENT

ESTIMATED K–RATIO GRADIENT

FIG. 2.41


k = AVERAGE HORIZONTAL STRESS / VERTICAL STRESS
0 1 2 3 4 5 6 7 8 9 10 11 12
0 100 200 300 400 500 600 700 800 900 1000 1100 1200 1300
DEPTH (m)
400 LEVEL
1000 LEVEL
2000 LEVEL
3000 LEVEL
4000 LEVEL
VARIATION IN k (HOEK–BROWN,1981)
LEGEND
● STRESS MEASUREMENT CSIRO GAUGE
+ STRESS MEASUREMENT USBM GAUGE
▲ STRESS MEASUREMENT CANMET

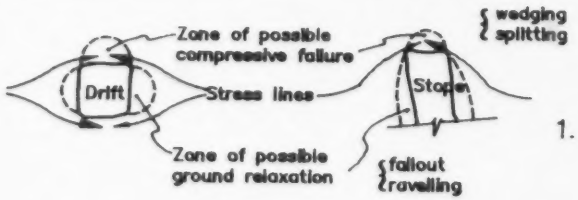
ESTIMATED K–RATIO GRADIENT

Stress State

Condition	Description
No Damage Evident	Walls of area generally sound, structurally controlled failure only.
Slight Stress Damage	Some slabbing of walls of area not evidently controlled by structure; drill holes non-circular after a period of time; give direction of long axis.
High Stress Damage	Area has blasted appearance, some difficulty in withdrawing drill steel. Dogearing of drill holes.
Failed	Area generally heavily fractured and of a loose nature (slack), considerable difficulty in drilling due to shearing and sloughing in drill holes.

TYPICAL FAILURE MODES

1. APPROXIMATE STRESS DISTRIBUTION

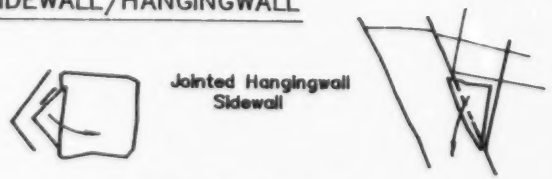


2. ROOF/BACK FAILURES

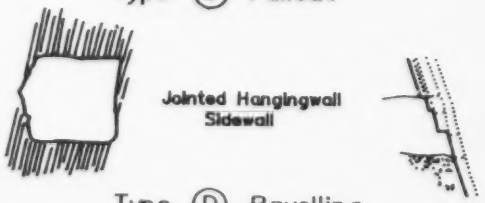
Type (A) Wedging

Type (B) splitting

3. SIDEWALL/HANGINGWALL

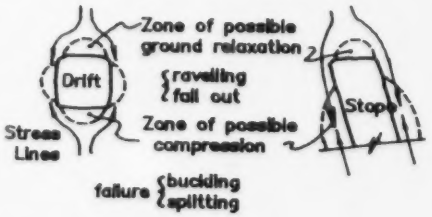


Type (C) Fallout



Type (D) Ravelling

1. APPROXIMATE STRESS DISTRIBUTION



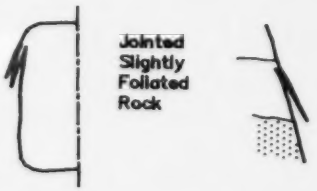
2. ROOF/BACK FAILURES

Type (1) Faliout

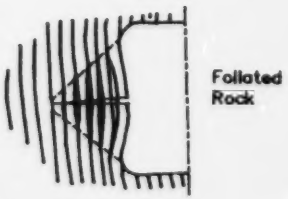
Type (2) Ravelling

3. SIDEWALL/HANGINGWALL

Type (3) Splitting



Type (4) Shear



Type (5) Buckling

FIG. 2.43

Scaling Effort

Condition	Description
Minor Scaling	General scaling of back and walls to remove small loose (<1 ft. maximum dimension.
Moderate Scaling	Several large blocks of loose (1 ft. to 5 ft. maximum dimension) exists.
Heavy Scaling	Numerous large blocks of loose exist. Ground working, need constant rescaling in order to secure.

SUPPORT TYPE

FIG. 2.45

Support Type

Condition	Description
0	None
1	Spot bolting
2	Pattern bolting – mechanical bolt
3	Pattern bolting – friction bolt
4	Pattern bolting – cable bolt
5	Pattern bolting plus shotcrete
6	Timber sets or cribs
NOTE:	A postscript of M or S to the category number will indicate the addition of mesh or straps respectively.

Support Load

Category	Description
0	No visible signs of load
B1	Bolt washer convex
B2	Bolt washer flat
B3	Bolt washer concave
B4	Bolt failed
B5	Bolt loose in hole
T1	Post and beam intact, wedges tight
T2	Posts splitting
T3	Beams splitting
T4	Post and beam splitting

OBSERVATION SHEET – OPENINGS

FIG. 2.47

LOCATION: _____ SHEET ___OF ___

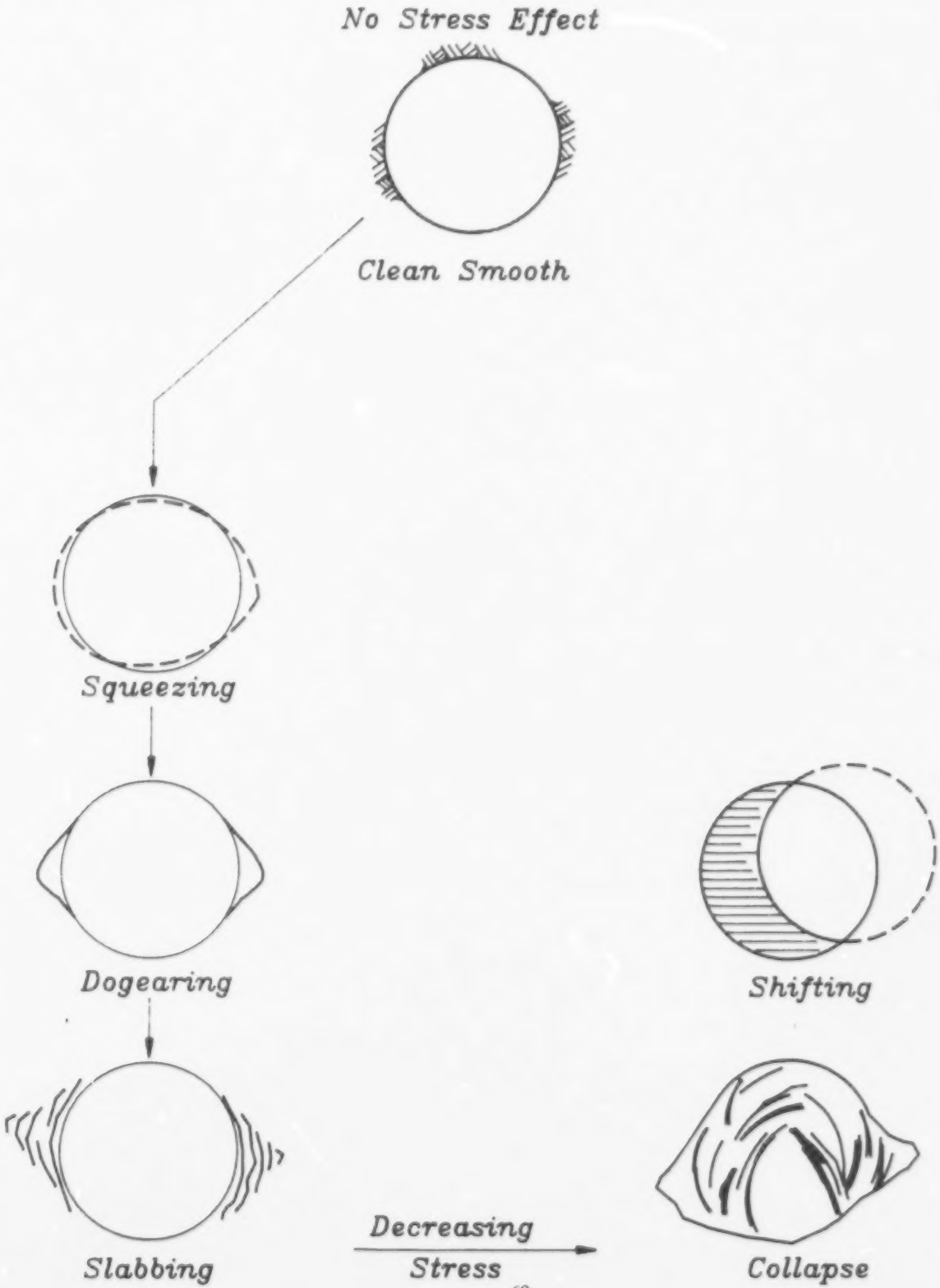
DATE: _____ LEVEL: _____ STOPE: _____

No. ON PLAN: _____ DISTANCE FW DRIFT TO STOPE: _____

EXCAVATION CONDITION:		STRESS STATE					SUPPORT		COMMENTS
EXCAVATION	SURFACE	LOOSE	NO EVIDENCE	SLIGHT	MODERATE	SEVERE	TYPE	LOAD	
FOOTWALL HAULAGE	HW								
	FW								
	BACK								
STOPE	HW								
	FW								
	BACK								
	ENDWALL								
DRILL DRIFT	HW								
	FW								
	BACK								
EXTRACTION DRIFT	HW								
	FW								
	BACK								

BLASTHOLE CONDITION

FIG. 2.48



ITH OBSERVATION

FIG. 2.49

I.T.H. OBSERVATION

DATE: _____

LEVEL _____ W.P. _____ BLOCK# ____ ROW# ____ HOLE DIA. _____in.

	1	2	3	4	5	6	7	8	9	10	11	12	13
BLAST DATE _____													
BLAST TYPE _____													
BLAST No. _____													
HOLE POS. TO FACE ___FT.													
VERTICAL													
HORIZONTAL													
HOLE COND.: CLEAN/SMOOTH													
OVAL													
STRESS													
SHEAR													
COLLAPSE													
TOP SILL: GOOD													
LIFTED													
CLOSED HOLES: REDRILL													
POUND OPEN													

LEGEND

(1) HANGING WALL SIDE

(2) STRIKE OF W.P.

EAST

(3) CRACKS

TYPICAL DRAWPOINT RECORDING SHEET

FIG. 2.50

FRAGMENTATION SURVEY

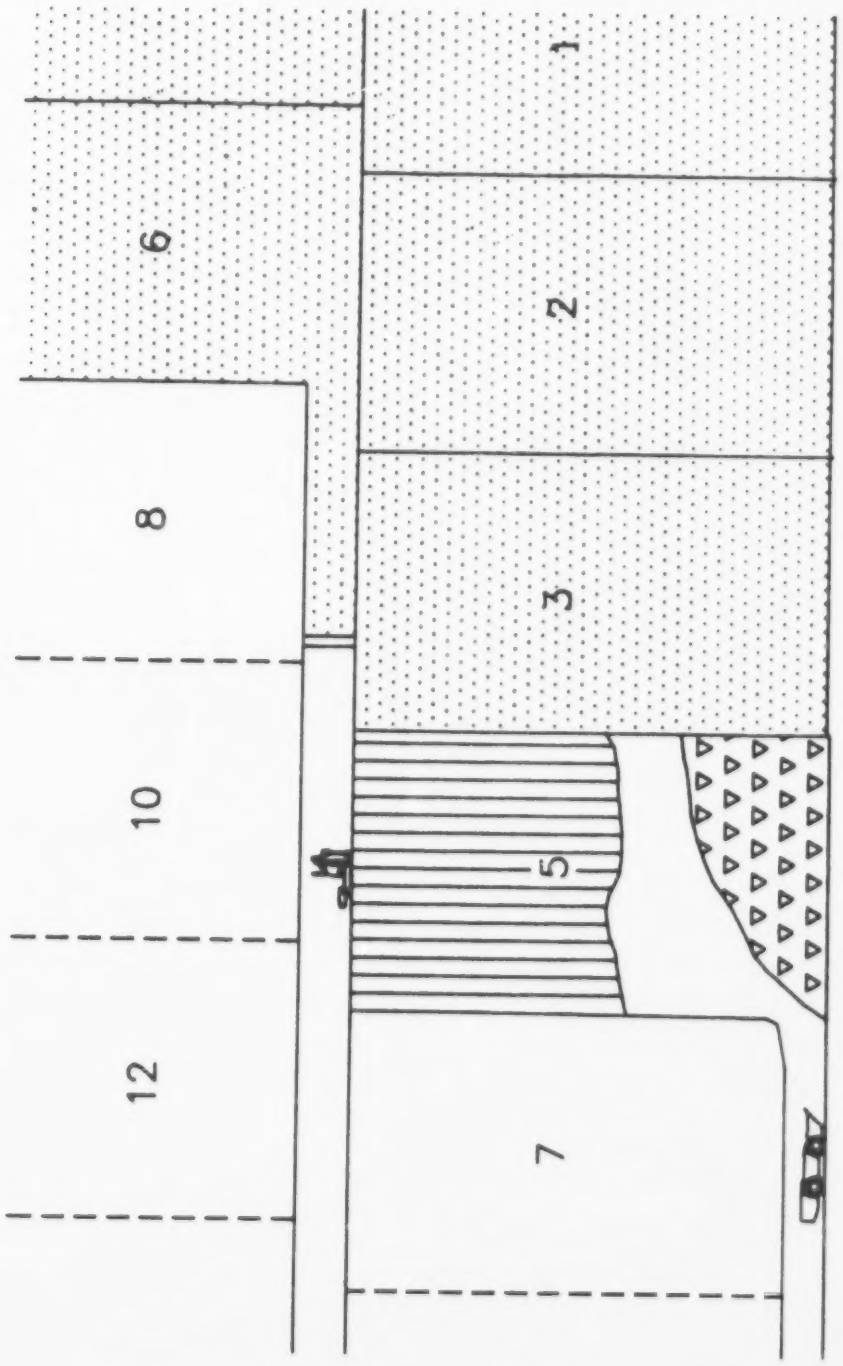
LEVEL _____ DATE _____

WORKPLACE _____ RECORDED BY____

DESCRIPTION	0	10	20	30	40	50	60	70	80	90	100	O	R	S
SULPHIDE														
< 6 "														
6 in – 1 ft														
1 ft – 2 ft														
2 ft – 3 ft														
3 ft – 6 ft														
6 ft – 10 ft														
> 10 ft														
0 JOINTS														
1 JOINT														
2 JOINTS														
3 JOINTS														
CUBIC S.F.														
TABULAR S.F.														
ROD S.F.														

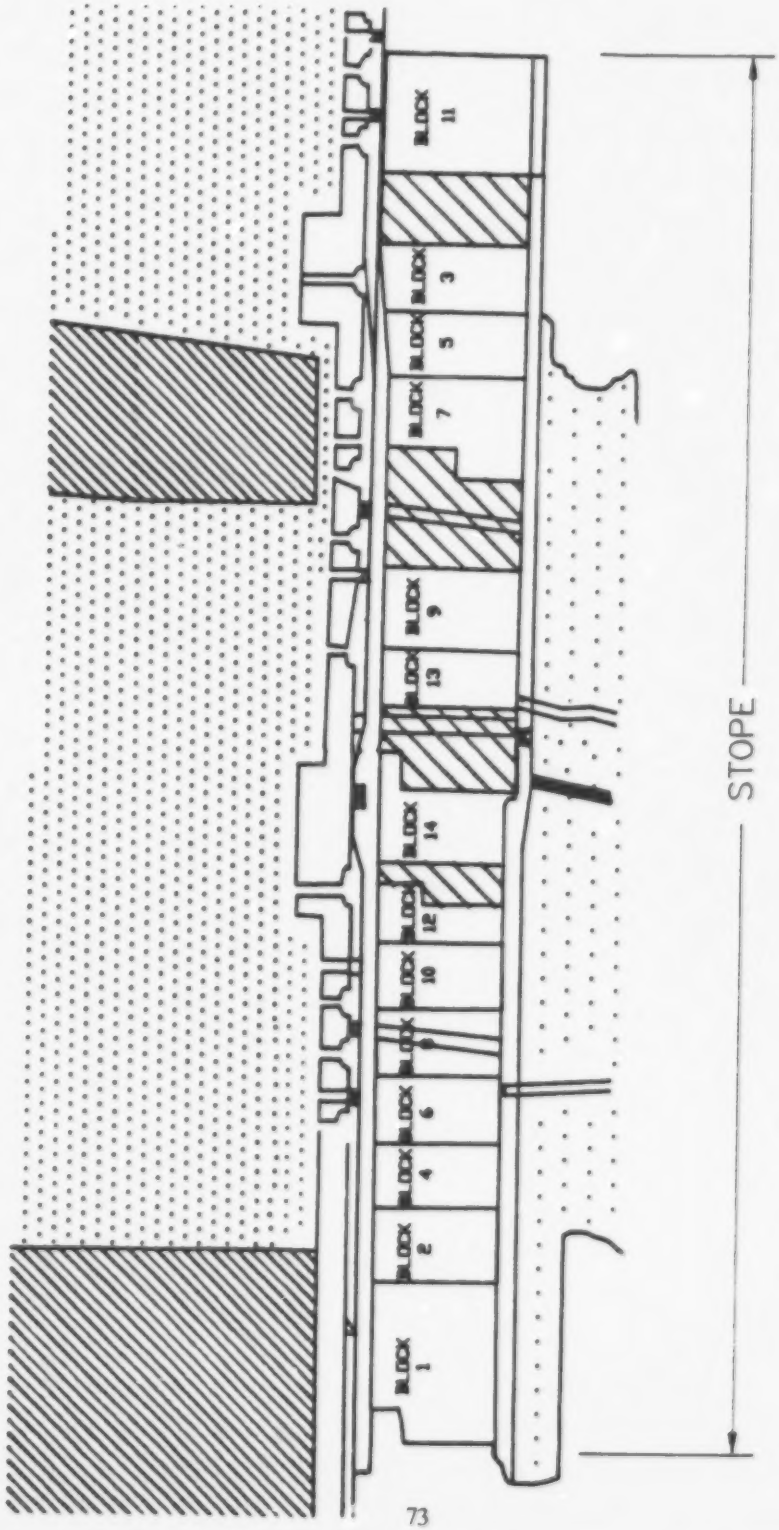
STANDARD VBM SHOWING STAGES OF MINING

FIG. 2.51



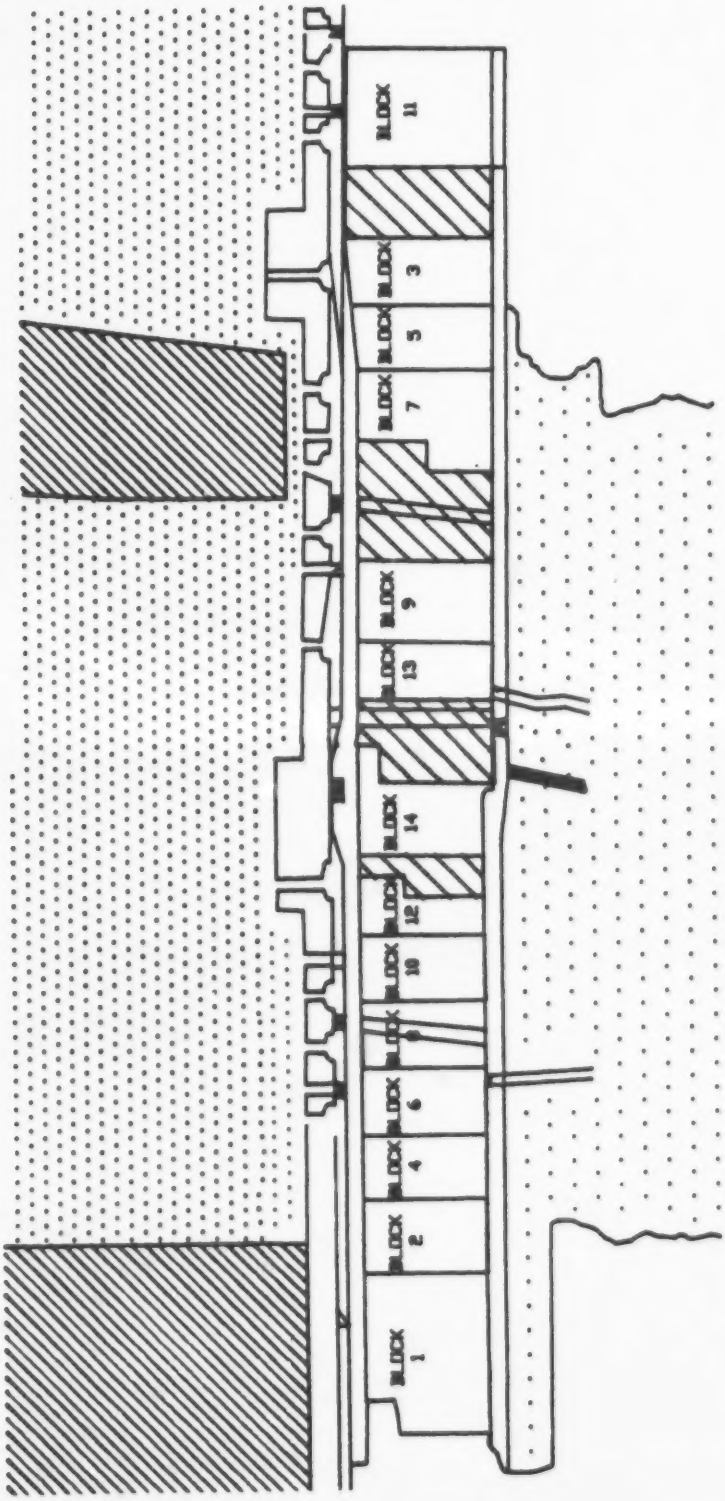
VBM BLOCK VS. STOPE

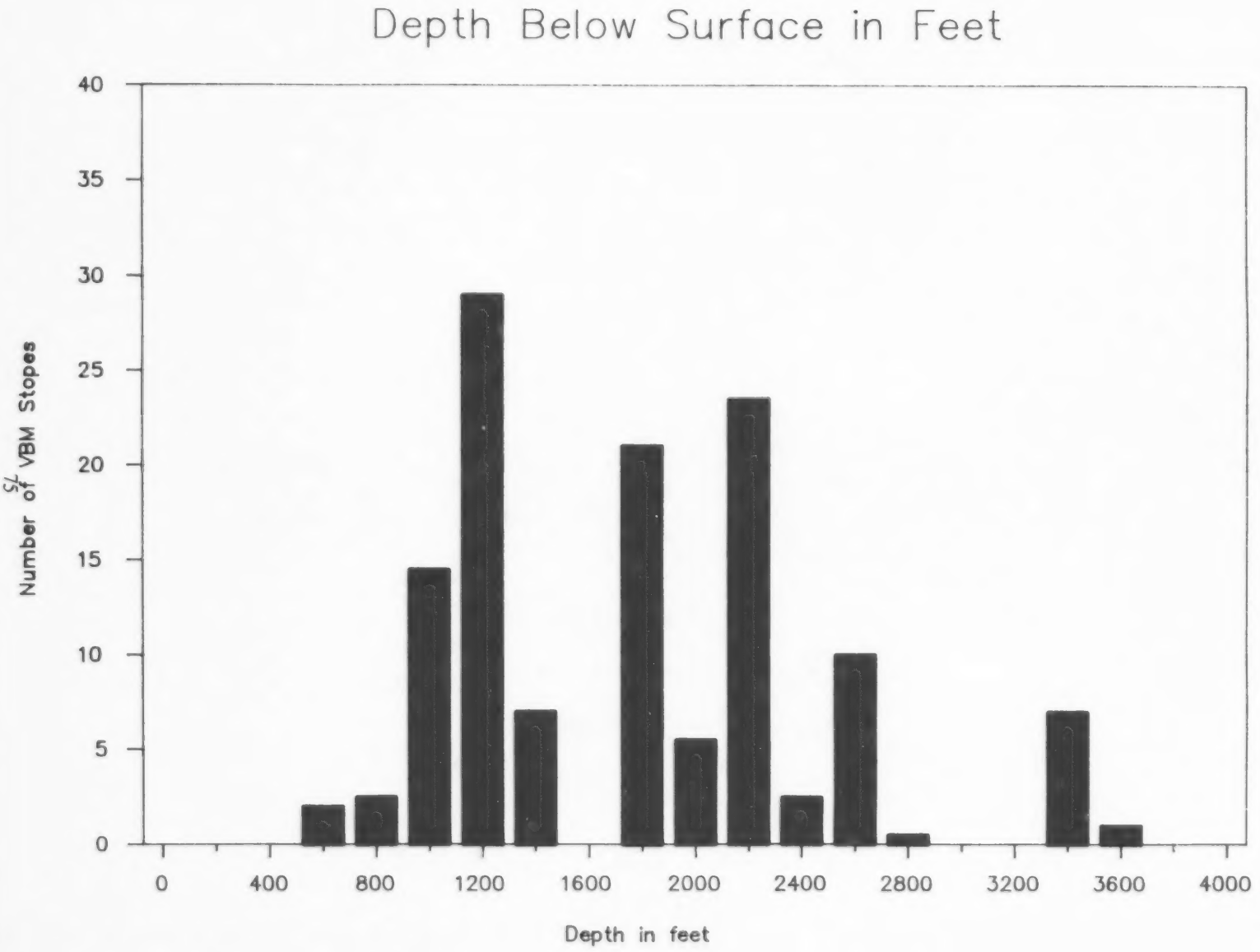
FIG. 2.52



RETREAT VBM–LONGITUDINAL MINING

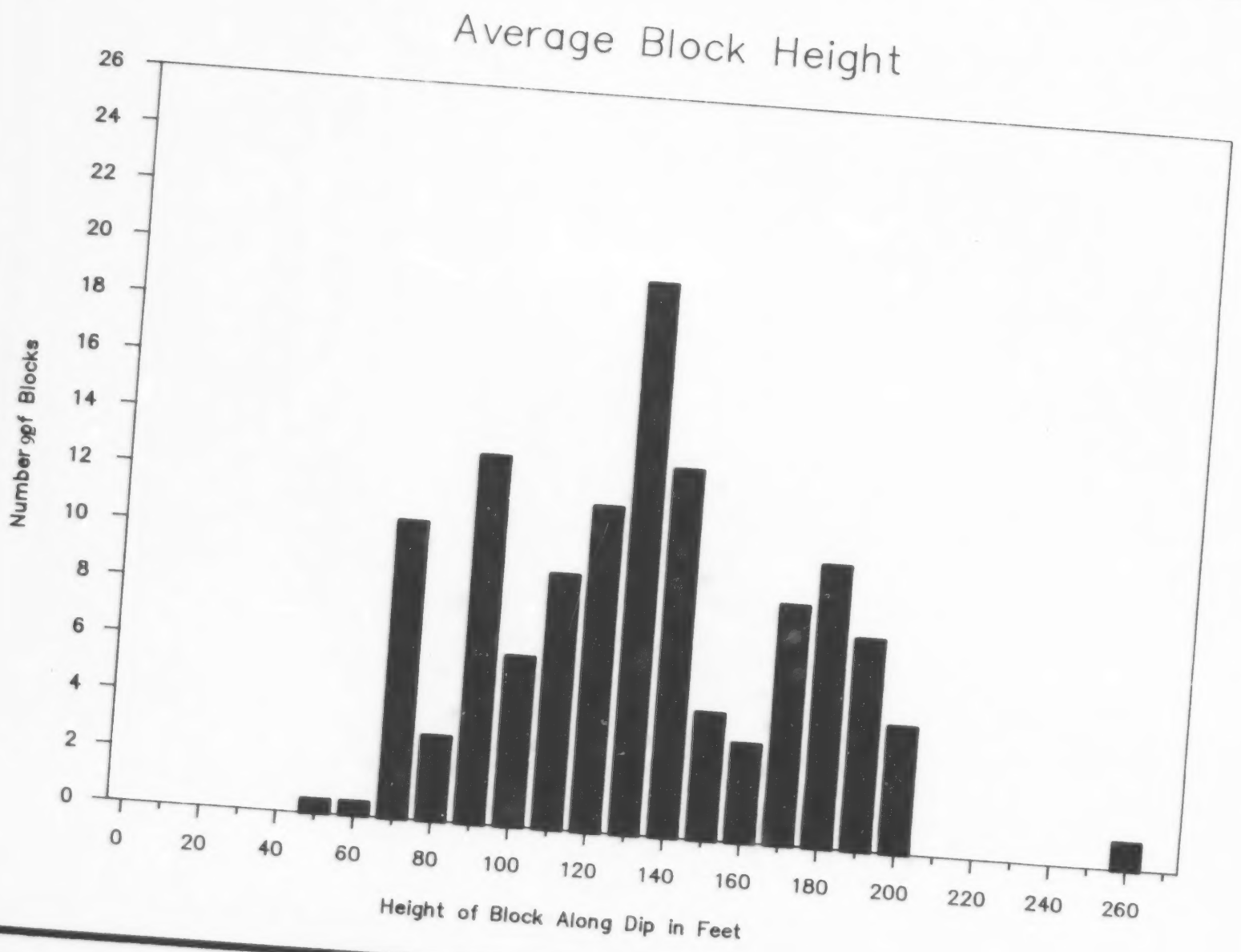
FIG. 2.53





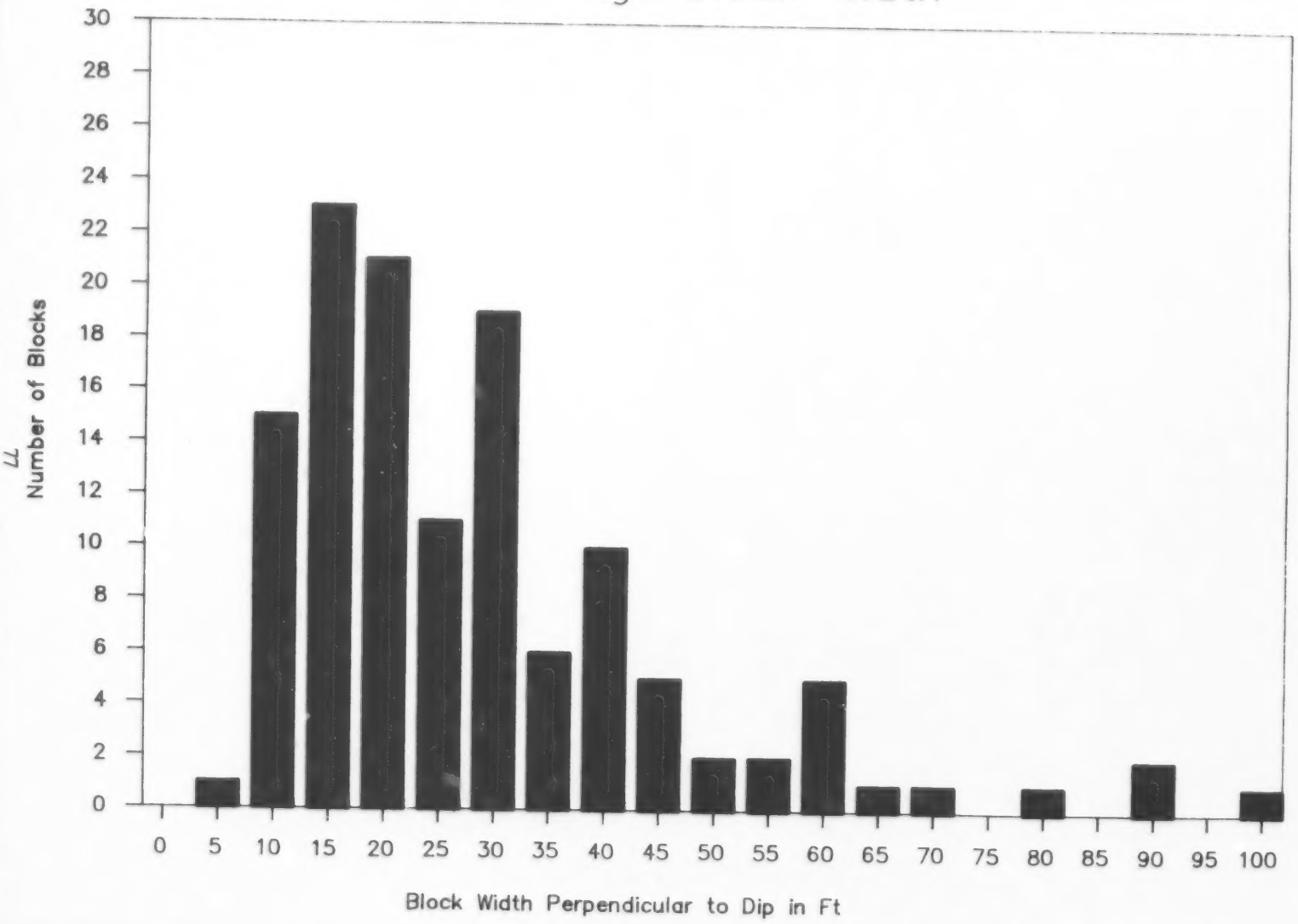
LOCATION OF VBM BLOCKS IN THOMPSON MINE

FIG. 2.54



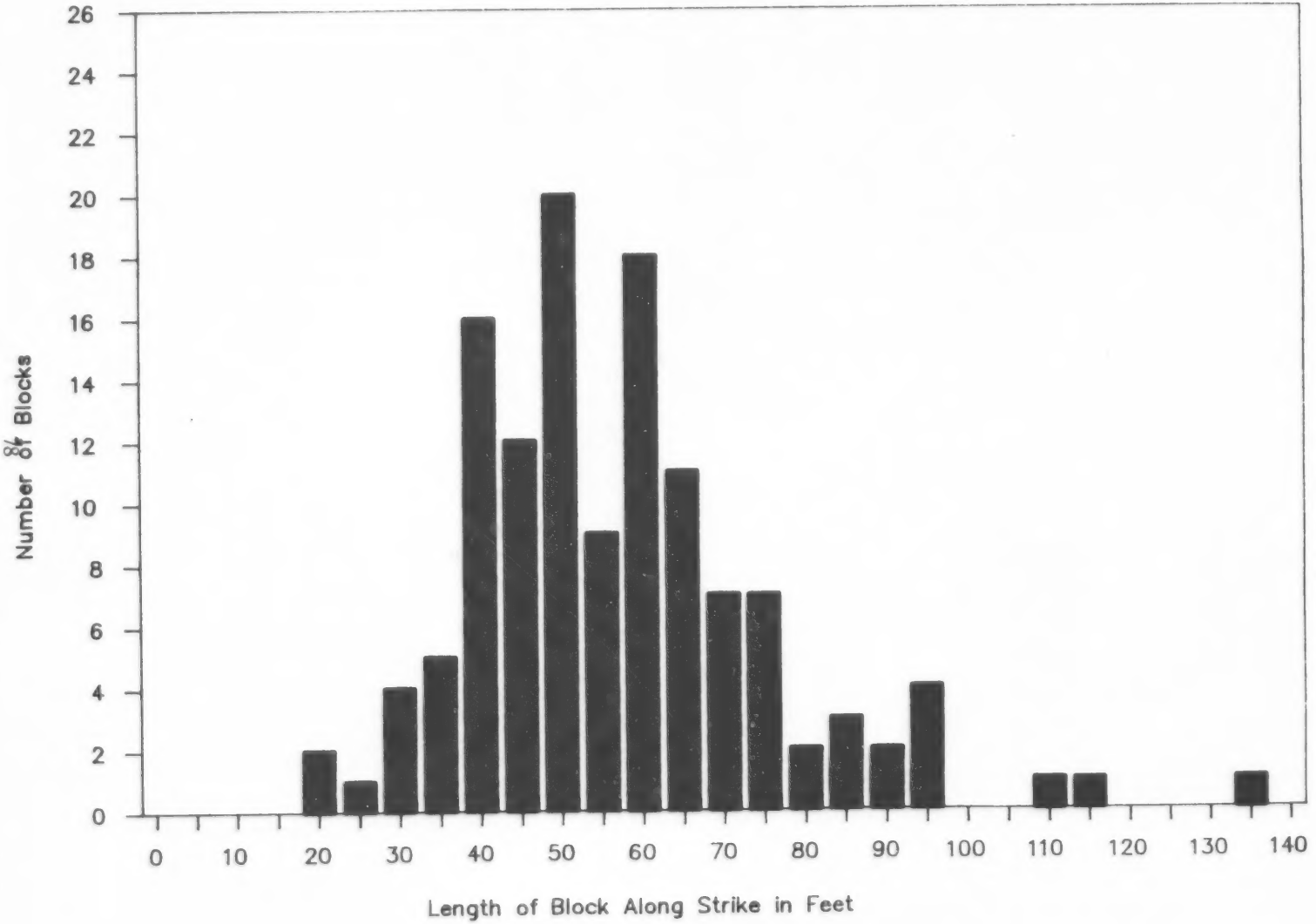
HEIGHT OF VBM BLOCKS AT THOMPSON MINE

FIG. 2.55



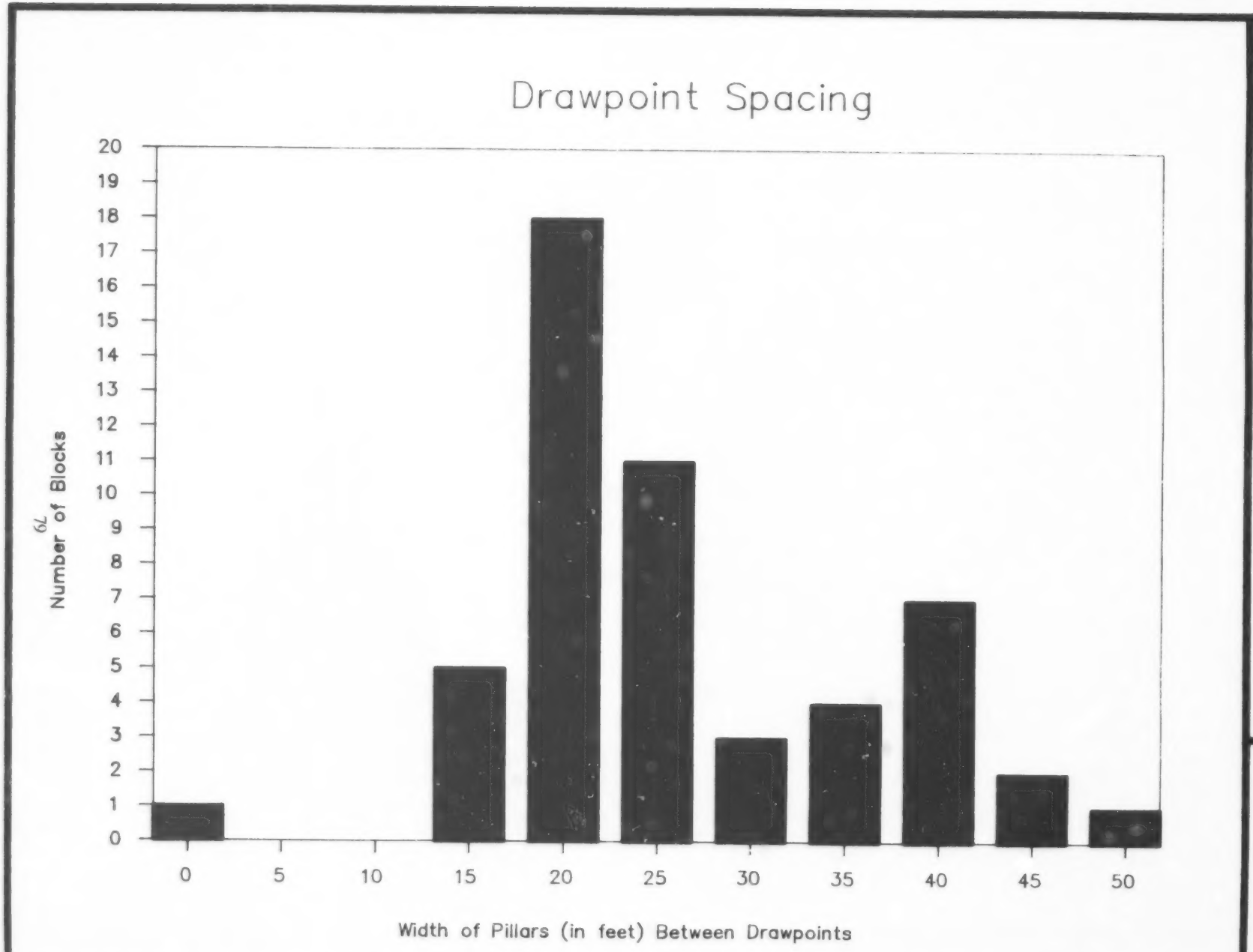
VBM BLOCK WIDTHS AT THOMPSON MINE

FIG. 2.56



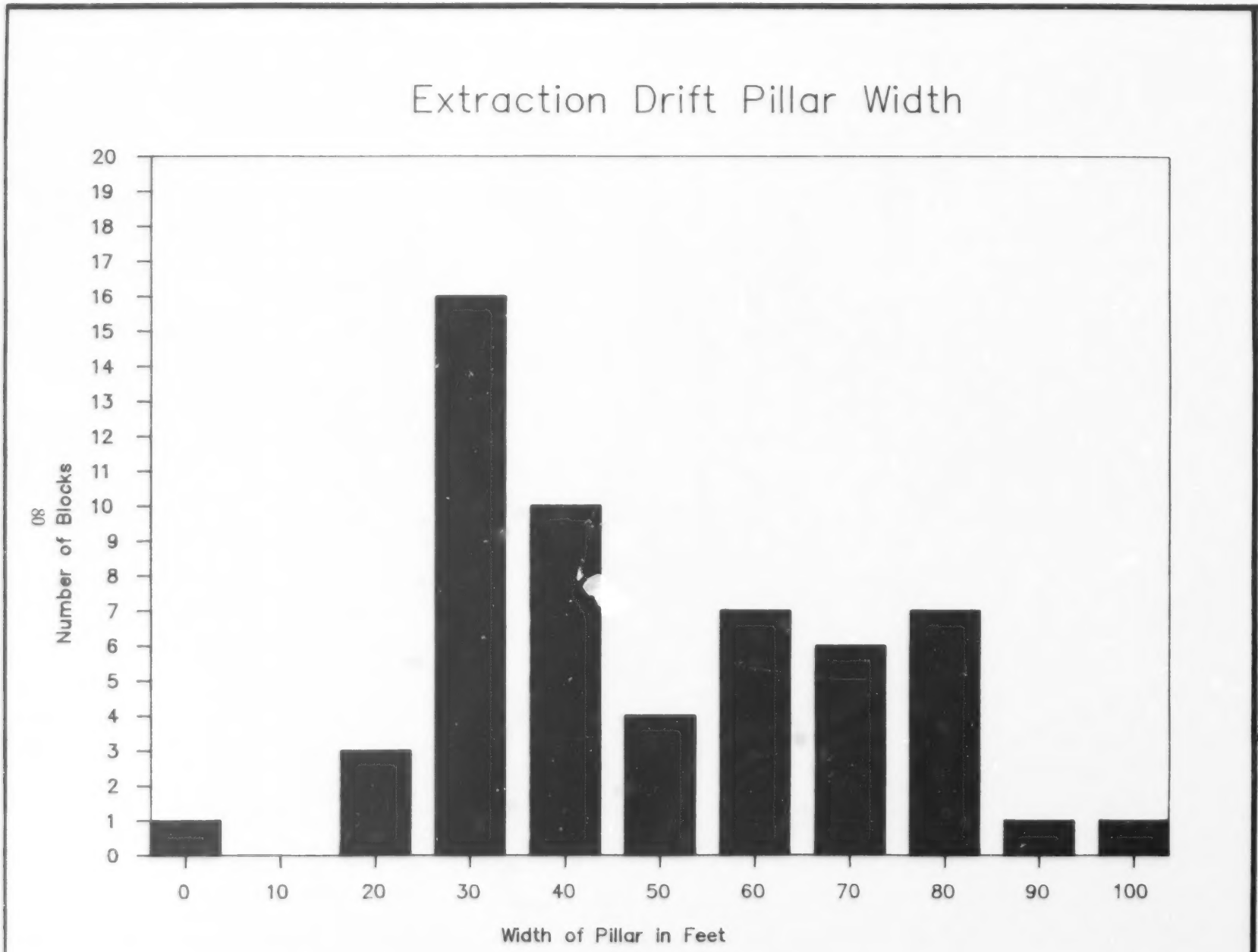
LENGTH OF VBM BLOCKS AT THOMPSON MINE

FIG. 2.57



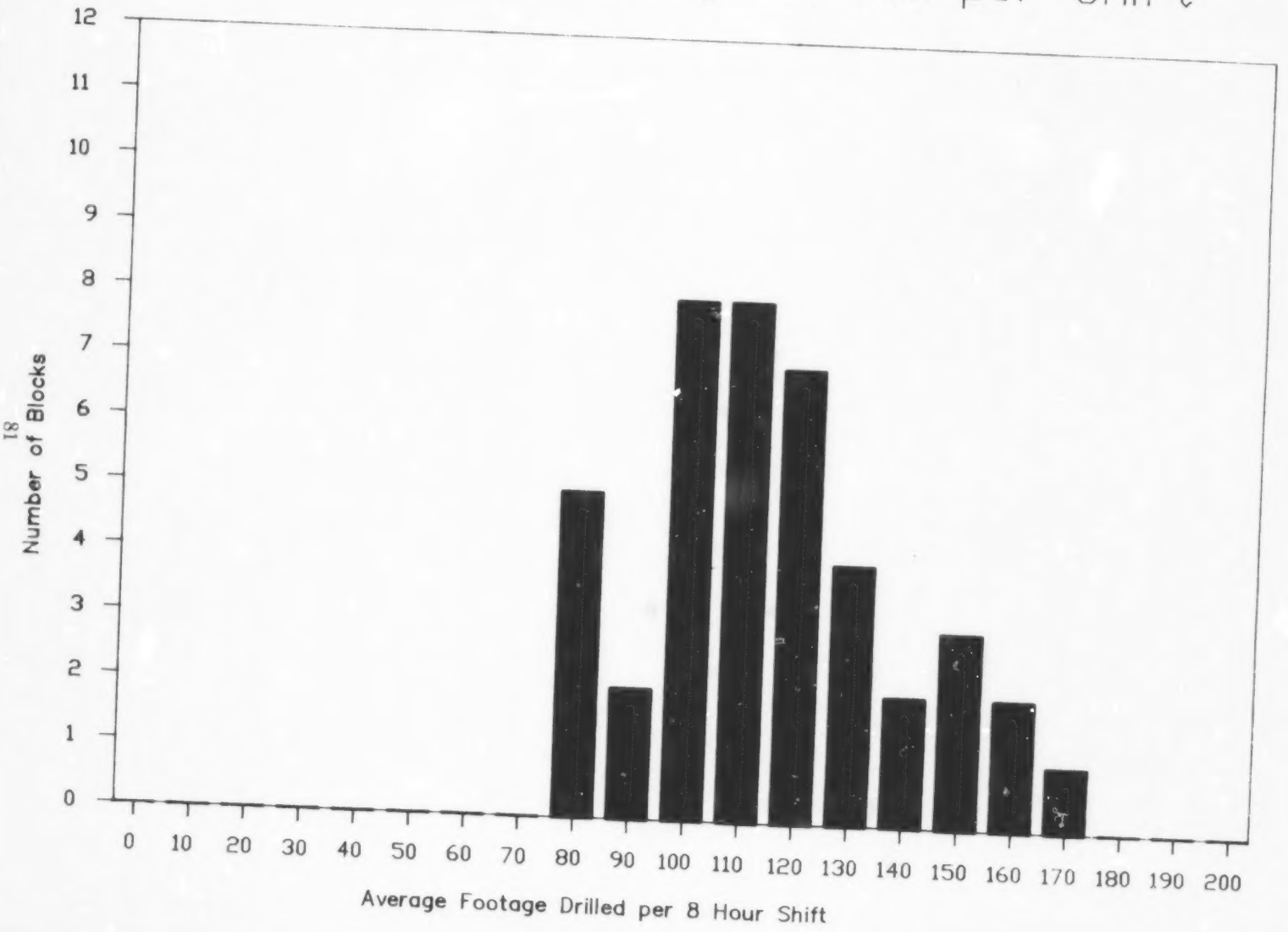
TRANSVERSE BLOCK DRAWPOINT SPACING

FIG. 2.58



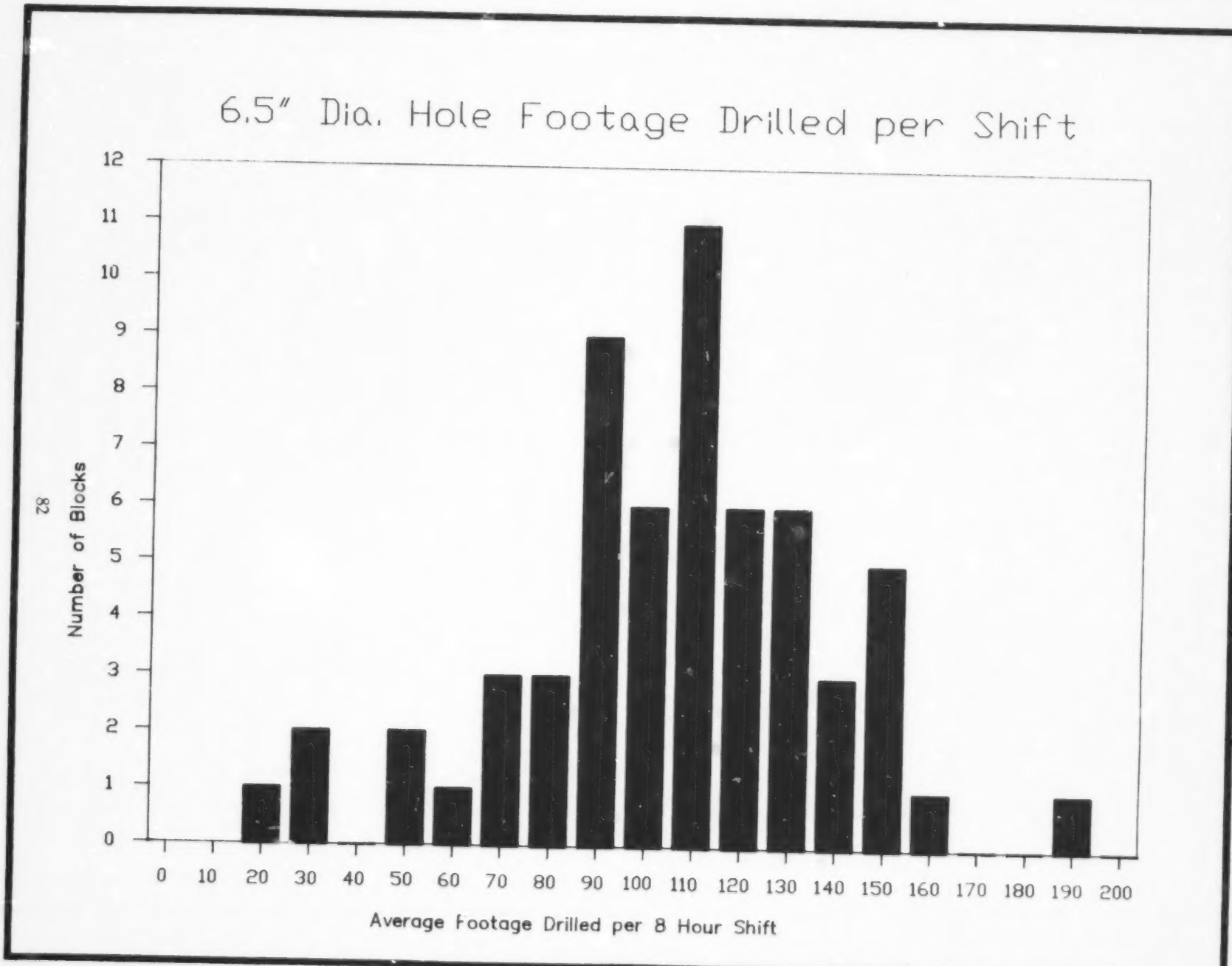
EXTRACTION DRIFT PILLAR WIDTH

FIG. 2.59



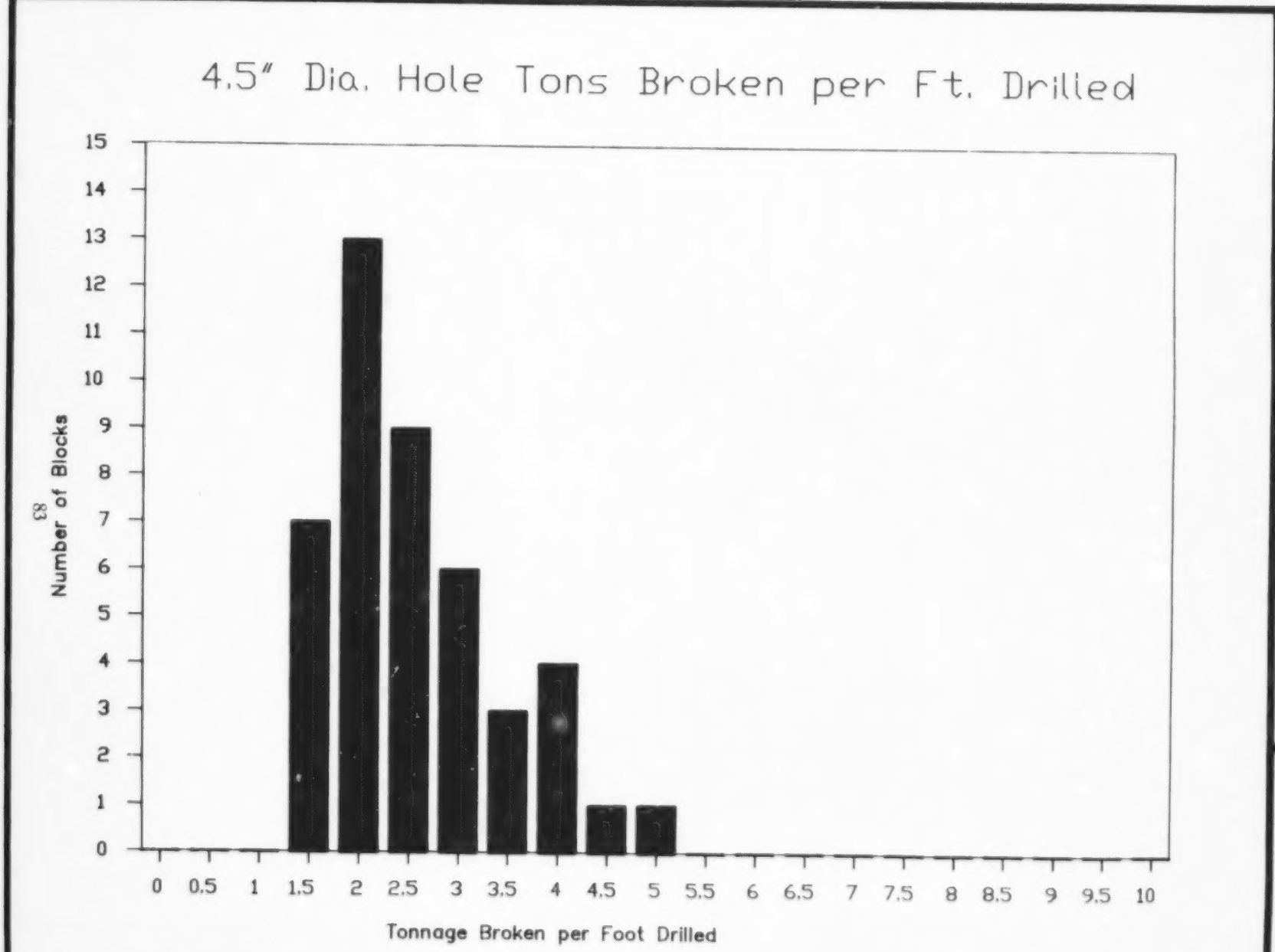
4.5" FOOTAGE DRILLED PER SHIFT

FIG. 2.60



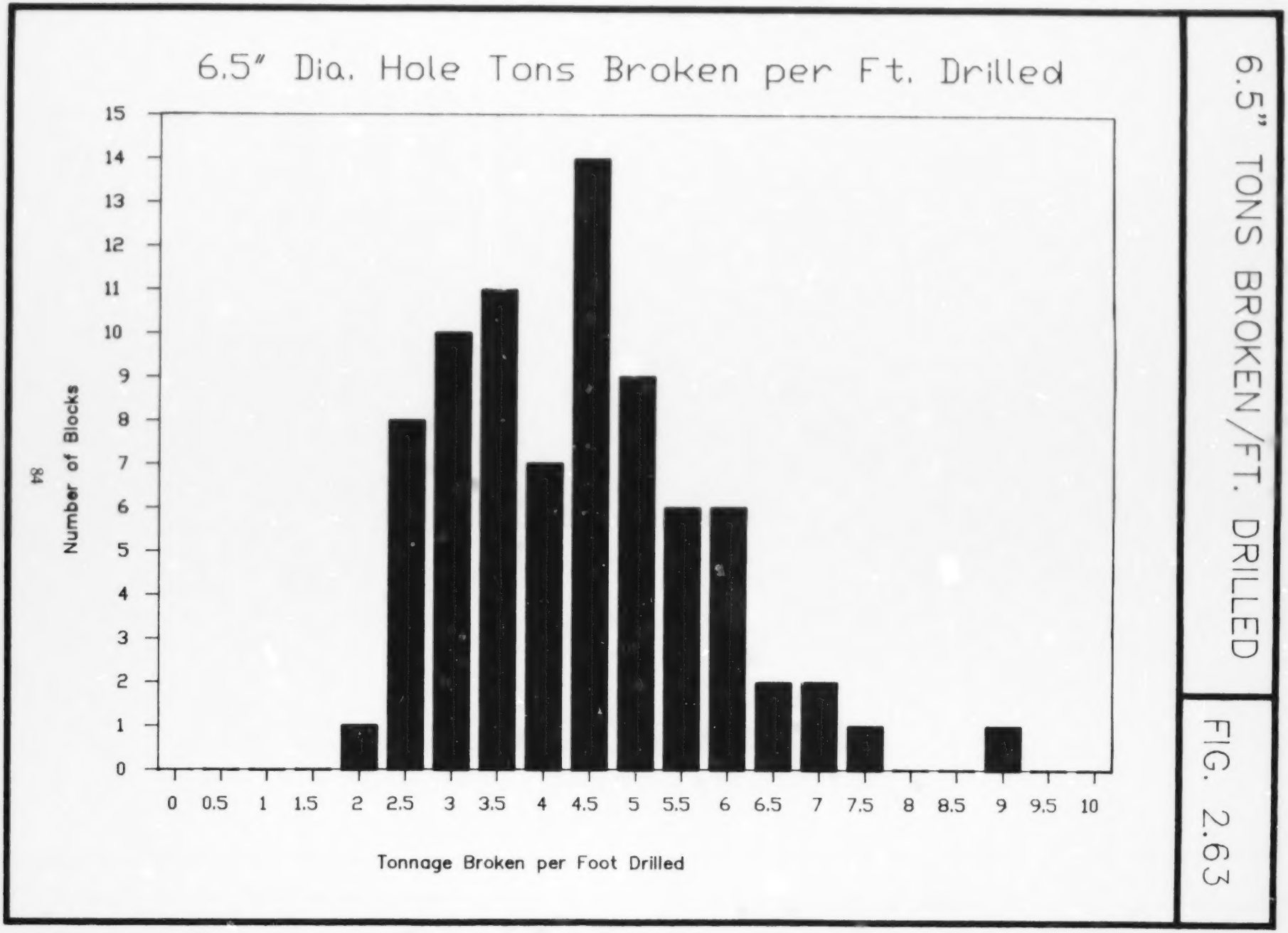
6.5" FOOTAGE DRILLED PER SHIFT

FIG. 2.61



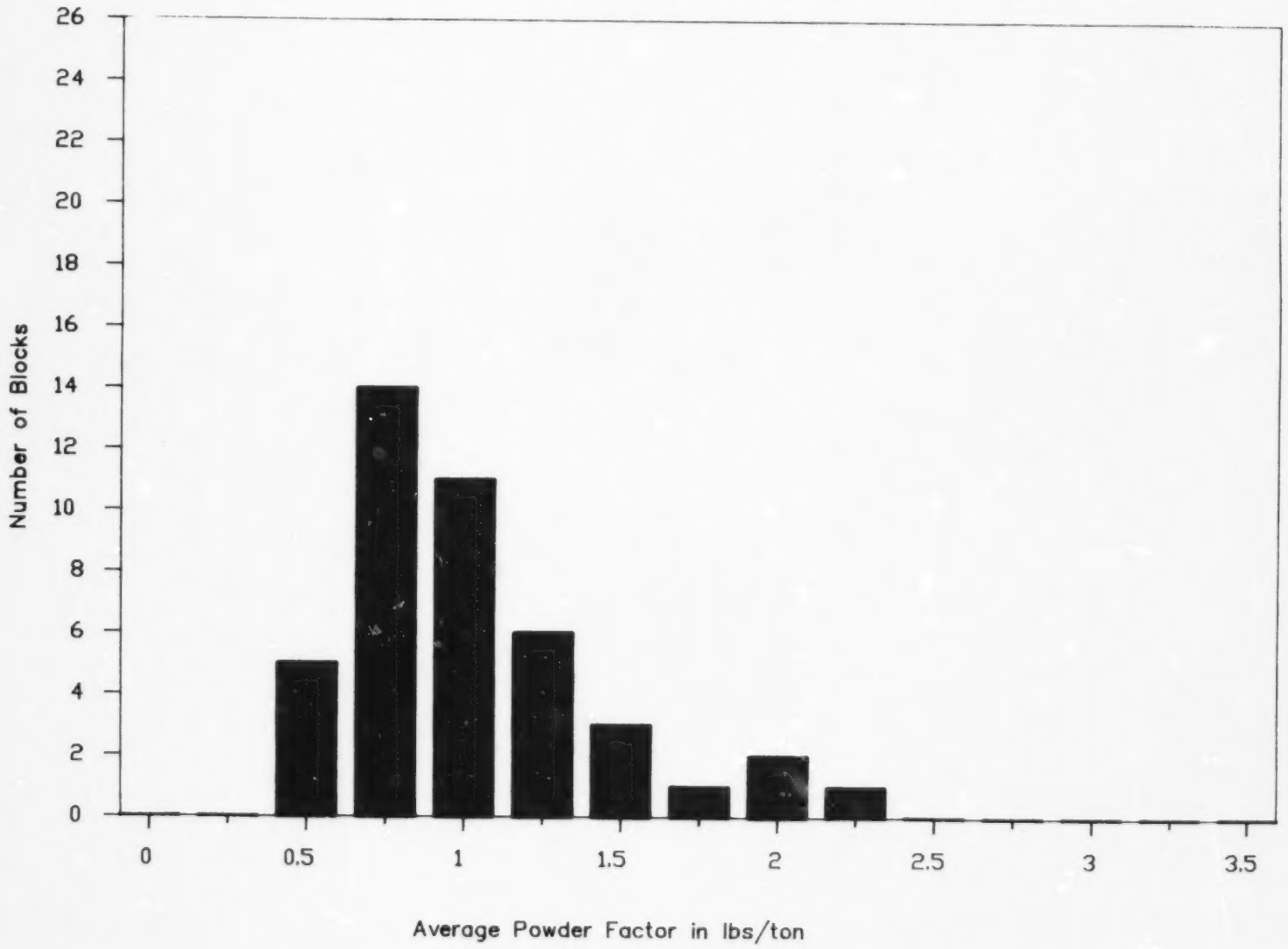
4.5" TONS BROKEN/FOOT DRILLED

FIG. 2.62



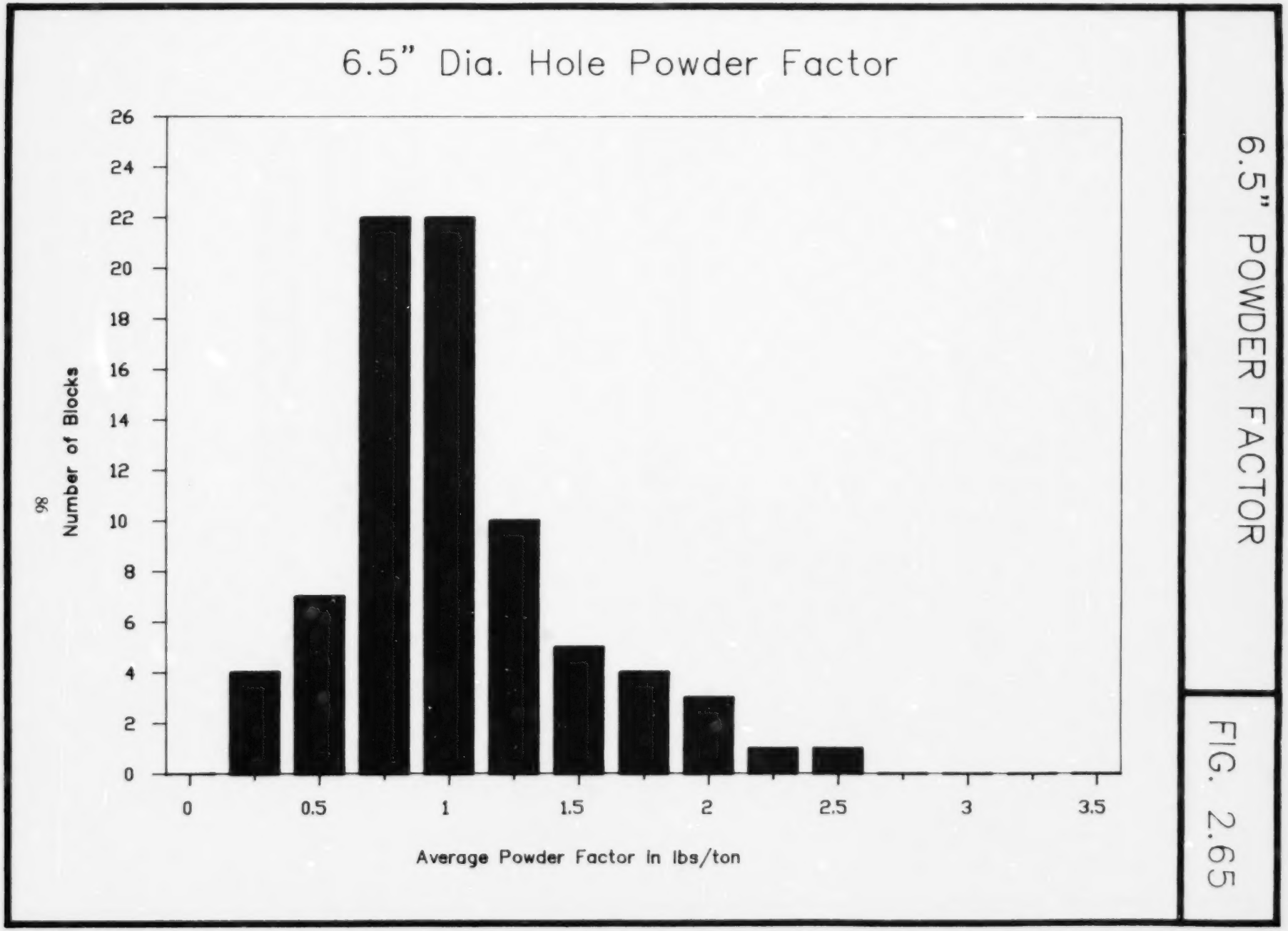
6.5" TONS BROKEN/FT. DRILLED

FIG. 2.63



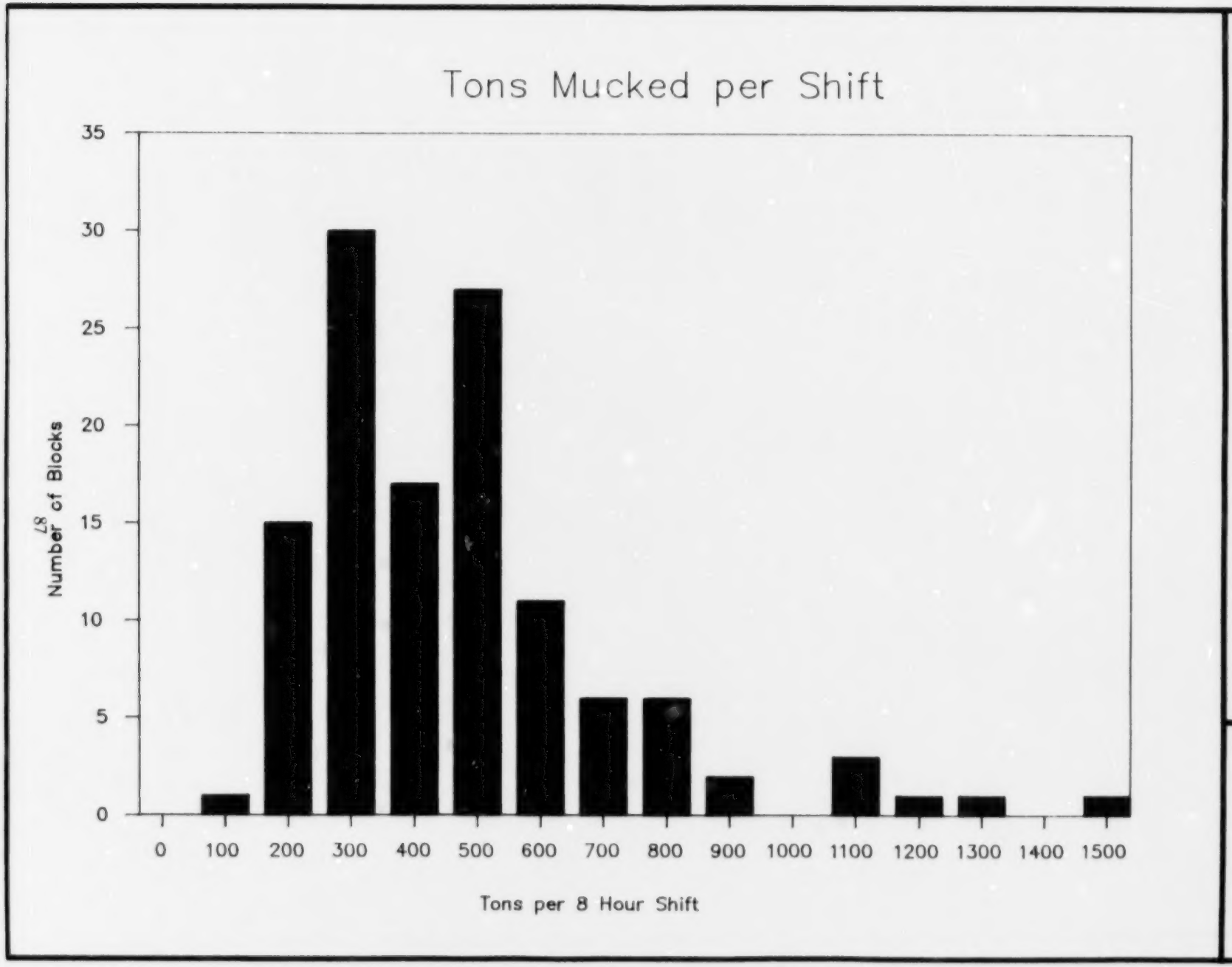
4.5" POWDER FACTOR

FIG. 2.64



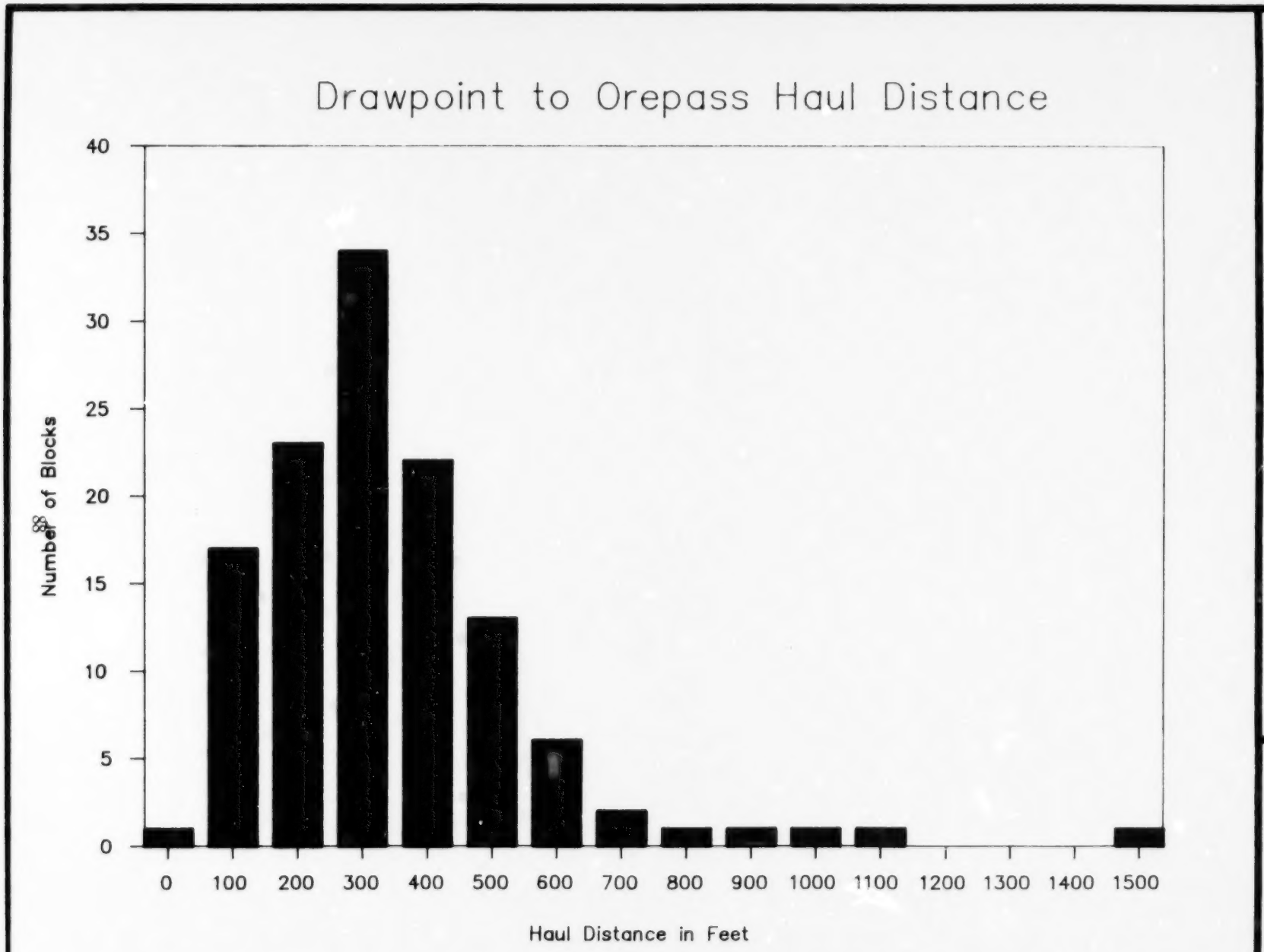
6.5" POWDER FACTOR

FIG. 2.65



TONS MUCKED PER SHIFT

FIG. 2.66



DRAWPOINT TO OREPASS HAUL DISTANCE

FIG. 2.67